Metallurgical and Chemical Engineering

An Incorporation of

Electrochemical and Metallurgical Industry—Iron and Steel Magazine

Vol. XII. No. 6. New York, June, 1914. 25 cts. per Copy. $2.00 per Year to U. S. and Mexico. $2.50 per Year to other Countries.

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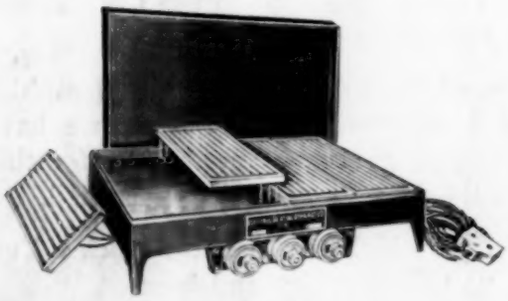

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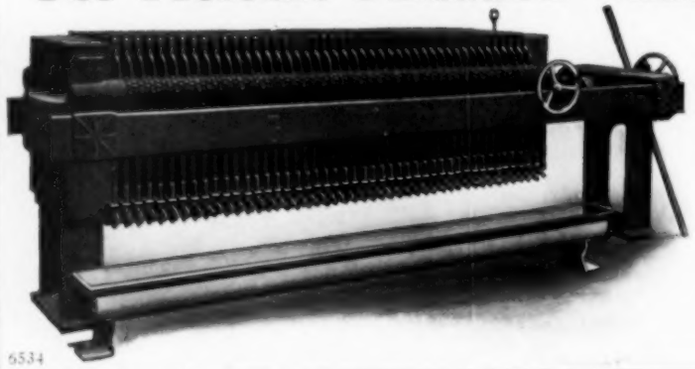

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Metallurgical and Chemical Engineering

A Consolidation of ELECTROCHEMICAL AND METALLURGICAL INDUSTRY and IRON AND STEEL MAGAZINE

VOL. XII — NEW YORK, JUNE, 1914 — No. 6

PUBLISHED MONTHLY BY THE

McGraw Publishing Company, Inc.

JAMES H. MCGRAW, President.
C. E. WHITTLESEY, Secretary and Treasurer.
239 West 39th Street, New York.

TELEPHONE CALL, 4700 BRYANT.
CABLE ADDRESS, METCHEM, NEW YORK.

WESTERN OFFICE.....421 Boston Bldg., Denver, Colo.
CHICAGO OFFICE.....Old Colony Building
PHILADELPHIA OFFICE.....Real Estate Trust Building
LONDON OFFICE.....Hastings House, Norfolk St., Strand

E. F. ROEBER, PH.D., Editor.
H. C. PARMELEE, Western Editor.

J. M. MUIR, Manager.

Yearly subscription price for United States, Mexico and United States dependencies, $2.00; for all other countries, $2.50. (European exchange, 10 shillings, 10 marks, 12.50 francs.)

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Entered as Second-Class Matter at the Post Office at New York, N. Y., under the Act of Congress, March 3, 1879.

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Paul Heroult

Out of the international group of the few great men who are the creators of modern industrial electrochemistry, Paul Héroult has gone into eternity. A sketch of his life is given elsewhere in this issue, but no such sketch can indicate the elemental greatness of the man whose loss we mourn. Héroult could have become a great mechanical engineer, but destiny made him a pioneer of electrochemistry. People wonder why Héroult and Hall should have found the same process of reducing aluminium simultaneously and independently in France and America. They might just as well wonder why it rains simultaneously in two places, if the air is oversaturated with water vapor in both. Humanity was ripe for the birth of industrial electrochemistry and the intellectual atmosphere was oversaturated with the ideas of its possibilities. Destiny's problem was to find the men who would be the nuclei of condensation. Héroult was one of them and in the history of mankind he will live forever as the pioneer of the electrometallurgy of aluminium and steel.

But in destiny's hands it is the man as man that counts. And those who knew Héroult best, loved him most as mere man. For ourselves we can say no more than that this journal has lost in him one of its oldest and truest friends. For all of his many friends all over the world we know we can say that though Héroult has gone, the memory of his genial personality will remain an everlasting inspiration.

Iron and Steel Output in 1913

With the appearance in the past fortnight of additional statistics of steel production in 1913 some interesting comparisons can be made. The 1913 statistics as far as reported are embodied in the following table, the figures referring to gross tons except in the case of nails:

	1912.	1913.	Change, Per Cent.
Pig iron	29,726,937	30,966,152	+ 4.16
Rails	3,327,915	3,502,780	+ 5.26
Structural shapes	2,846,487	3,004,889	+ 5.55
Rods	2,653,553	2,464,807	— 7.12
Wire nails (kegs)	14,659,700	13,559,727	— 7.50
Cut nails (kegs)	978,415	842,038	—13.90

The outstanding feature in the comparison is the decline in rod production. While rods are used for the production of chain, the great bulk of the output is consumed in the manufacture of wire so that the decline in rod tonnage may be regarded as a decline in the production of wire. The production of wire nails, it may be noted, decreased in substantially the

same ratio as the production of rods. The greater decrease in cut nail output represents no new trend, for the cut nail output has decreased nearly every year since the maximum was reached in 1886, or since the wire nail began to come into vogue.

Fifteen years ago the statisticians were making predictions that eventually the production of rods would exceed the production of rails. It was an interesting thing to say, and the bare figures appeared to furnish adequate support. The production of rods had increased from 279,769 tons in 1888, the first year for which statistics were gathered, to 1,071,683 tons in 1898, an increase of 282 per cent. Rails had made the poor showing of being unable to surpass the tonnage made as far back as 1887. Those statisticians, however, overlooked certain things. Fully one-third of the 1898 production of rods went into nails, and this represented quite largely a displacement of the cut nail by the wire nail, a displacement then almost completed. They overlooked also that wire products were purchased and consumed by the people for their ordinary daily requirements, hence the five-year industrial depression from which the country was emerging had not very seriously curtailed wire consumption. On the other hand, the railroads were very poor. A large percentage were just emerging from receiverships. In 1899, a year of great activity in iron and steel, the rod production actually decreased, whereas rail production made a new record, at last passing the tonnage of 1887. From 1899 to 1906 rail production increased 78 per cent while rod production increased 84 per cent., rods, therefore, making hardly any gain over rails. Since 1906 there has been some good reason, however, for repeating the old prediction, since from 1906 to 1913 rail production decreased 12 per cent., whereas rod production increased 32 per cent., and in 1913 rods amounted to 70 per cent., as large a tonnage as rails.

Reverting to the comparison between 1912 and 1913, it is remarkable that rails and structural shapes made gains slightly greater than were made in pig iron, and rods, a light product, showed a loss, for the steel trade philosophy of recent years has been that the gains in tonnage were likely to be made by the light products, particularly in years in which financial conditions were not especially favorable to the launching of big enterprises. These big enterprises involve the consumption of structural shapes very largely, and it is commonly held that in no year since 1906 have conditions been nearly as favorable as in that year for the financing of such things. Nevertheless, we find that the production of structural shapes increased 41.9 per cent., while the production of pig iron increased only 24.3 per cent. Given such an expansion in new construction and the launching of big enterprises as is to be expected some day, the tonnage of structural shapes promises to show further large gains. In 1913 they fell only 14 per cent. under rails, when 15 or 20 years ago their tonnage was almost insignificant.

Sweeping Decision in Flotation Litigation

The United States Circuit Court of Appeals for the ninth circuit, sitting at San Francisco, handed down on May 4 a decision in the case between James M. Hyde and Minerals Separation, Ltd. The latter had brought suit against the former as infringing Minerals Separation flotation patents, and had secured a favorable decision in the lower court. The Court of Appeals reversed this decision and upset Minerals Separation patents on concentration by flotation.

The decision is of wide interest and has been awaited impatiently by many metallurgists who were concerned in one way or another with the future of flotation in this country, and who viewed with apprehension the prospect of a monopoly in that new and promising branch of metallurgy.

The language of the court is forcible, clear-cut and unequivocal and the point of view on which the decision is based is interesting. The basis of novelty and invention claimed by Minerals Separation lay in the formation of a froth containing concentrates produced by agitation of pulverized ore in water, by the action of oil in quantity less than 1 per cent of the ore treated. The froth and the minute quantity of oil were the main points. The court reviewed the patents in the art beginning with those of Haynes (British) in 1860, and Everson's (U. S.) in 1886, and found each of the steps in Minerals Separation's process "fully described in more than one of the patents of the prior art, with the single exception of the reduced quantity of oil." The mere formation of a froth by agitation was held to be anticipated in the prior art.

The attitude of the court regarding the small quantity of oil used in the Minerals Separation process was that reduction in the amount of oil required for flotation was only a logical development in metallurgical economy and efficiency such as any progressive concern would bring forward in the course of its investigations. The mere use of a smaller quantity of oil than in other processes is, therefore, held not to be sufficient to constitute invention, for this difference is one of degree and not of kind. To hold such a claim valid would be "to take from others the right to use oil economically." Quoting from another decision the Court said: "A change in form, proportions or degree, doing substantially the same thing in the same way by substantially the same means, with better results, is not such invention as will sustain a patent"; and on its own behalf it held "that to sustain the appellee's (Minerals Separation) patent would be to give the owners thereof a monopoly of that which others had discovered."

There are two courses yet open to Minerals Separation before it may consider its case finally lost. It may seek a review of the decision by the U. S. Supreme Court directly or it may endeavor to get a different decision in another circuit of the Court of Appeals and if successful, bring the case before the Supreme Court for final judgment. Either course is

beset with very great difficulties. It may be expected, however, that Minerals Separation will not accept defeat until it has exhausted every possibility of carrying the case to the highest court, just as it has done successfully in its Australian litigation. It is equally true that it will find ready adversaries along the way, eager to break the monopoly which undoubtedly has deterred some big operators from openly adopting flotation.

However, with the patent statutes once definitely and finally declared, we believe the flotation process will be widely used in this country and prove especially valuable in reducing mineral waste.

High-Frequency Spectra of Elements and the Structure of Matter

It is well known that when a substance, which for definiteness may be a simple elementary substance, such as aluminium, is powerfully heated, it emits radiation with a certain range of variation in regard to frequency or wave-length. Such wave-lengths may extend from about 0.1 μ to 30 μ or more, the symbol μ designating a micron, or the millionth of a meter. The range of the visible spectrum is known to be roughly from 0.4 μ to 0.8 μ.

When an elementary substance is supported in a vacuum-tube and is made a target for electric discharges, the substance emits X-rays. For a long time it was supposed that X-rays had no wave-length; but recent researches have shown that they consist of two types of radiation, one of which is irregular or non-rhythmic, so as to have no wave-length; while the other is regular and rhythmic, possessing a wave-length. This X-ray wave-length is, however, very much shorter than that of the ordinary incandescence spectrum. It may range from 60 $\mu\mu$ to 800 $\mu\mu$ where the symbol $\mu\mu$ is a micro-micron, or the millionth of a millionth of a meter. It is only possible, at present, to measure these extraordinarily small wave-lengths, by means of reflecting mirrors of molecular type. That is to say, the polished reflectors of ordinary light are far too coarse-grained, and crystalline surfaces of natural cleavage have to be used.

In recent numbers of the Philosophical Magazine, Mr. H. G. J. Moseley has described some important experimental researches on the frequencies and wave-lengths of X-rays emitted from targets of different substances employed successively as the cathode in an X-ray tube. The generated X-rays, after passing through a small slit, are allowed to fall upon the surface of a crystalline reflector. The incident beam of X-rays emerges from the crystalline surface in certain definite directions, when the angle of incidence is just right for rays from the particular substance under investigation. The reflected beam is caught on a photographically sensitive plate, and leaves a record line on its surface after the plate has been developed.

The whole experimental system has ordinarily to be worked in vacuo, as the air would absorb the faint reflected beam.

It is shown that each elementary substance has a definite X-ray frequency of emission, such that, if the square-root of the frequency is taken as abscissas, the various elementary substances plot along a straight line in order of atomic weights. Thus, every element from aluminium to gold, possesses a certain whole number N, which determines its X-ray spectrum, and which Mr. Moseley calls the "atomic number" of the element. It is noticeable that whereas the atomic weight of an element is not a whole number or integer, the X-ray atomic number is claimed to be in all cases a simple integer, commencing with 13 for aluminium, and ending with 79 for gold, with only three gaps, which are suggested to belong to elements as yet undiscovered. Moreover, the atomic number N is identified with the number of positive units of electricity in the atomic nucleus of the atom. The order of succession of the atomic numbers is, with a few exceptions, the same as the order of atomic weights, and the exceptions appear to exist at points in the series where the chemical properties of the elements disagree with the order of atomic weights. Expressed in another way the squares of the successive numbers are proportional to the X-ray spectrum frequencies of the various elementary substances.

The observations also indicate that in the elements investigated, the angular momentum of an electron in any elementary atom is the same. That is, the product of the square of the orbital radius of an electron, into its angular velocity about the center of the atom, is the same for all electrons and atoms. If this result can be verified, it will constitute a remarkable generalization concerning the constitution of matter. It supports the hypothesis that each negative electron pursues an orbit about the positive atomic nucleus, and is held in equilibrium at that orbital radius where the inwardly directed electric attraction balances the outwardly directed contrifugal force. The attractive force on each planetary electron would thus be a purely electrostatic force, enormously greater than that of gravitation; while the repulsive force would be a purely magnetic force, not differing except in magnitude from the force of inertia in the world immediately perceived by the senses.

There is thus presented experimental evidence for the belief that the atoms of different elementary chemical substances differ fundamentally, and in the most simple order of classification, by the number of positive electrons forming their respective nuclei. Each atom may be regarded as a sun-and-planet microcosmic system, in which the positive nucleus corresponds to the sun and the gyrating negative electrons to the planets. Whereas in the cosmic universe, planets differ in size and in angular momentum, in the microcosmic or atomic universe, planets all have the same size, and also the same angular momentum. Oscillations in the orbital paths of electrons due to cathode-ray impacts, give rise to the high-frequency spectrum oscillations of the atom; whereas oscillations between molecules produce the lower-frequency spectrum oscillations that give us the colors of the rainbow.

Readers' Views and Comments

A Suggestion of a New Method for Determining the Temperature of Ingots

To the Editor of Metallurgical & Chemical Engineering:

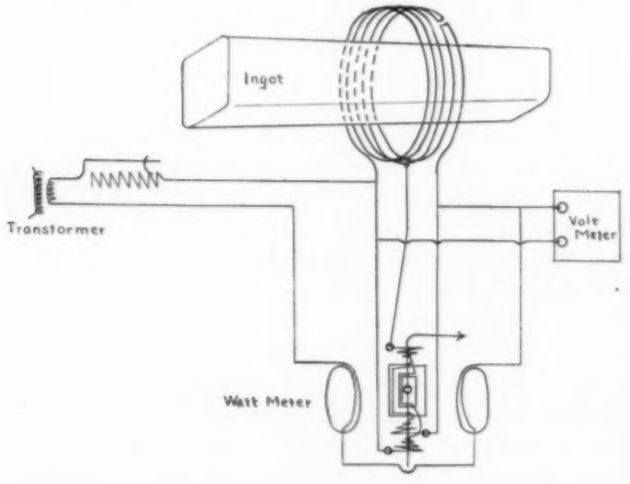
SIR:—Considerable saving in power required to roll steel and other meals could be made if a suitable means were devised to control the temperature of the object to be rolled. No doubt improved quality of the finished metal might also result if uniform methods could be practised.

In the quenching of steel objects such as tools, permanent magnet forms, etc., valuable data might be obtained if the time-temperature curve were readily obtainable.

The value of an idea which might result in the accomplishment of satisfactory temperature determination, is the excuse for these remarks.

The determination of temperature of a steel ingot, for instance, is accomplished best at present probably by the use of thermocouples, though in certain cases it might be possible to use the electrical resistance method.

Using the thermocouple it is practicable to apply



ELECTRIC METHOD FOR DETERMINING TEMPERATURE OF INGOTS

same to the ingot surface only, but this surface is covered with an oxide coating more or less variable in thickness so that the indications produced by the thermocouple are hard to correct for average temperatures within the ingot. Certain methods of heating ingots would also influence the correction factor to be applied to the surface indications; the centre of the ingot may or may not be hotter than the surface. Considerable time is required when a thermocouple is used due to its low heat conductivity as a rule. Heat is also abstracted from the place of contact.

The resistance method requires contacts with the heated object, which are difficult to obtain as desired; the method is slow also therefor.

Electrical resistance, magnetic hysteresis and permeability are functions of the temperature. Suppose a coil be provided (as indicated in the sketch), through which an alternating current was caused to flow, the e.m.f., wave form, frequency, etc., being maintained constant. The ingot whose temperature was required might be introduced into the field of this coil and the sum of the eddy current and hysteresis energy losses read from the wattmeter. The I^2R loss of the coil could be automatically subtracted from the total energy loss so that the wattmeter would indicate energy manifestations within the ingot only. A differential wattmeter might be used as indicated, and this procedure should obviate the error which would otherwise result due to resistance change of the coil with heat.

The indication of the wattmeter would be instantaneous and could be readily obtained in temperature figures with a suitable calibration curve.

The calibration curve might be absolute or empirical, but certainly it would obtain only for a given ingot of fixed size and practically fixed composition. The position of the coil with respect to the ingot would also require to be practically fixed.

This method would require no contacts with the heated object other than those of the charging machine or manipulating device ordinarily used.

JOHN G. HOMAN.

Follansbee Brothers Company,
Follansbee, W. Va.

Power Problems in Electrochemistry

To the Editor of Metallurgical & Chemical Engineering:

SIR:—I have been much interested in reading the papers on the power problems in the electrolytic deposition of metals in your February issue, page 91, and your May issue, page 328.

Month	No. Days Operating	No. Hours Operated	No. Hours Shutdown	Cost of Shutdown (Interest & Depreciation)	Cost of Repairs	K.W.H. Generated	Cost of Operation per K.W.H. Coal Only	Cost of Fuel Charged per K.W.H.	Total Cost Per K.W.H.	Cost of Operation	Total Operating Charges for Month	Load Factor Based on Plant Capacity
Jan.	360	8617.7	$9920.00	$7617.72	$1109.75	2385,430	.0264	.0379	.0643	$1507.30	$9707.94	.719
Feb.	300	7241.	8068.75	6310.40	94.55	2044,230	.0261	.0361	.0622	1350.37	8710.39	.71
Mch.	324	7772.	8952.00	6631.60	97.40	2256,830	.0260	.0361	.0621	1349.05	8549.05	.665
April	324	7749.4	8921.00	6607.45	43.20	2190,700	.0256	.0344	.0600	1307.48	8257.48	.73
May	270	6320.	7280.00	5771.35	43.40	2264,700	.0213	.0362	.0575	1418.87	8213.87	.71
June	264	6340.	7304.00	6297.42	43.64	2142,000	.0297	.0388	.0685	1470.00	8990.00	.72
July	288	6912.	7960.00	6280.48	43.10	2264,000	.0255	.0360	.0615	1386.88	8008.38	.715
Aug.	306	7320.	8396.00	6656.83	44.10	2268,800	.0254	.0316	.0570	1248.86	8719.38	.60
Sept.	330	7920.	9120.00	6636.42	54.00	2280,000	.0273	.0277	.0550	1.547	8899.00	.7709
Oct.	348	8352.0	9620.00	6621.50	58.80	2321,510	.0260	.0290	.0550	1.38	8960.80	.712
Nov.	318	7632.0	8790.00	6596.58	51.30	2244,990	.0255	.0280	.0535	1.388	8818.68	.72
Dec.	288	6912.	8284.00	6569.68	54.00	2252,120	.0260	.0290	.0550	1.570	8652.18	.637
Total	3600	8644.1	99580.75	6606.60	644.81	23580770					14480.98	93480.98	.619
Aver. Per mo.	300.00	7197.	8298.39	6606.60	53.73	1965064	.0264	.0470	1.194	$1252.41	$8123.41	.72

RECORD OF ONE YEAR COMMERCIAL OPERATION

I am enclosing a blue print showing a recapitulation of a year's log sheet from an 800-kw gas power plant operating on bituminous coal that costs $3.50 per ton. By checking the various items in this table carefully you will note that with uniform load factor somewhere near the capacity of the plant and when in twenty-four hour operation power could be developed at a very low figure in such a plant. The cost data in this recapitulation are carefully kept, and all of the expenses, including fixed charges, incident to the operation of the plant, are figured in. I believe that such equipment as this could be advantageously used in the electrochemical industry. I note from the discussion of the various papers that there is sufficient waste heat from the reverbatory furnaces in these plants to furnish all the steam necessary for heating purposes. Considerable heat can also be recovered from the waste from the gas engines if the waste from the furnaces is not sufficient.

Regarding Mr. Vom Baur's comments on the possibility of oil engines to operate on Mexican fuel oil, I

will say that Diesel engines in Texas will not operate successfully on this fuel. This can be verified on the ground by any one interested in the subject. There are probably more Diesel engines in Texas than in any other part of this country and they are operating on high-grade oils.

R. M. CHATTERTON.

Smith Gas Power Co., Lexington, Ohio.

The Heat Treatment of Steel—Results Obtained by Spring Makers and Steel Works Treatment

To the Editor of Metallurgical & Chemical Engineering:

Sir:—The question of heat treatment is now attracting the attention of producer and consumer to a degree almost unparalleled in the field of steel manufacture. In order to get satisfactory results, it must be approached in a practical and scientific manner, because if the consumer carries his specifications to an extreme extent, it will be almost impossible to secure steel from the producers.

An extensive research was carried out by the author in 1910 upon a carload of steel that had been rejected by a prominent spring manufacturer, who claimed that it could not be "tempered" even in water. Upon receipt of the carload of steel, an analysis was made for carbon which was found by combustion to be 0.95.

The question arose at once whether there was some hidden cause, why the steel could not be tempered as the spring makers wrongly called it, whether there was something wrong with their methods or something wrong with the combustion apparatus and supplies used, whereby high-carbon results were being obtained. The last question was attacked first and it was finally found, by running checks upon known standards, that the apparatus, supplies and results obtained were correct.

The next problem was to answer the claim of the spring maker that this steel could not be "tempered," or rather hardened, as it should be called, in water. Several pieces were selected at random from the car, heated in a mill furnace and hardened by two methods in a bosh of water, both lots of steel being heated to a like degree. In the first method the steel was quenched as soon as it was drawn from the furnace, in the second method the steel was allowed to cool to a point slightly over the recalescence point and then quenched, this being done to approach as nearly as possible the method followed in some manufacturing plants.

In both cases by dropping the steel upon the mill tracks, it was found to be as brittle as glass. The first lot quenched had an extremely fine grain, and the second had a coarse structure on account of the crystals having time to grow, before the steel was quenched. This answered the statement that the steel would not harden in water.

Upon this report being turned into the office, it was put up to the writer to find out just what was wrong. It must be taken into consideration that this steel was an ordinary carbon steel and, therefore, any kind of treatment was good enough, from the viewpoint of the manufacturers.

The heat numbers shown in table 1 were selected for the extended research, several billets being rolled into 1¾ x ¼ square edge flats.

TABLE 1—HEATS SELECTED

Heat Nos.	Shippers Carbon	Works Analysis
1490	0.96	1.09
1493	0.92	0.90
1524	0.96	0.94
1527	1.00	1.05
1538	1.02	1.00
1540	1.12	0.94
1541	1.06	1.04
1550	0.90	0.90
2565	1.08	1.03
2582	0.90	0.90

Bars selected were marked Lot Nos. 1 and 2. Bars from Lot No. 1 were nicked on one end in order to make a fracture after tempering. In order that the test should be a practical one, a rolling-mill furnace was selected after it had cooled down from its day's run. The temperature was then brought up to the desired point, the bars put in and each one heated uniformly and quenched in oil, the furnace was then cooled down and the bars placed back and the temper drawn, until the oil just flashed.

It must be taken into consideration that this method was followed in order to duplicate as closely as possible the spring makers' methods, then employed.

These bars were then tested in an oil hydraulic spring tester having an upward thrust, they being placed on two knife edges on tester tables, the pressure being taken up in the center by means of another knife edge in head of tester. In order to get a sensitive measure of the permanent set, the bars had been formed with a camber of one-half inch.

Lot No. 1 was tested at a modulus of rupture of 130,000 lbs.; Lot No. 2, 140,000 lbs.

TABLE 15—RESULTS OF TESTS
130,000 lbs.

Heat	Lot One	Permanent Set	Fracture	Heat	Lot Two	Remarks
1490		None		1490		Lot. No.
1493	"	"	All	1493	"	one heat
1524	"	"		1524	"	2565.
1527	"	"	uniform,	1527	"	carried
1538	"	"		1538	"	up to a
1540	"	"	fine	1540	"	modulus
1541	"	"		1541	"	of 150,000
1550	"	"	and	1550	"	Permanent
2565	"	"		2565	"	set 1/16"
2582	"	"	silky	2582	"	

Loads applied 5 times.

Feeling safe after such a test, rollings were made from these heats and shipped. An awful holler was raised, it being emphatically stated that this steel could not be tempered by the regular shop methods and that a special treatment had to be designed.

After these reports were turned in, it was decided that the trouble was a case of six on one side and half a dozen on the other, both the steel and spring manufacturers being at fault.

The conclusion drawn from the research was that in order to get satisfactory results from any type of steel, carbon or alloy, it was necessary to give the steel a proper treatment. Also that the method used for manufacturing the steel leaves its effect upon the steel so produced.

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The Western Metallurgical Field

Carnotite Concentrators for Colorado

Essentially two methods of concentrating carnotite ores have been developed as the result of experimentation during the past few years. Ordinary methods of concentration based on difference in specific gravity of minerals will not suffice, for the uranium-vanadium mineral exists as a thin coating or sand grains and produces a slime or powder when removed from the gangue. Hence special methods of separation and concentration have been devised—both wet and dry. The general principle of these methods has been made public by the Bureau of Mines in Bulletin 70[1], and details of a wet process also have been published in this journal.[2]

Until recently, however, no attempt to concentrate these ores has been made on a commercial scale, although the economic reasons for so doing have always been recognized. The Standard Chemical Company, of Canonsburg, Pa., is to be the first in the

1. See this journal February, 1914, page 89.
2. Article by Siegfried Fischer, October, 1913, page 586.

Colorado district to build concentrating plants. This company owns a large number of claims and is probably the largest single producer of carnotite in this country. It has figured prominently in the "radium hearings" before congressional committees, and has been concerned over the proposal of the Government to produce radium; but with the Government's position made clear by Dr. Joseph A. Holmes,[3] undue anxiety on this score may now be allayed.

It is reported that the Standard company will erect several concentrating plants at different points in southwestern Colorado: one near Naturita, 60 miles west of Telluride; a second in the East Paradox district, and a third in the McIntyre field. A fourth may be built in Utah to handle ore from the southeastern part of that state. The immediate construction of the first mentioned is assured, and machinery for it has been purchased.

No announcement has been made as to the details of the process to be used, but the general principles can be surmised from the references given. It is known, however, that Mr. Warren F. Bleecker, now on the metallurgical staff of the company, developed concentrating processes for carnotite while making independent researches on uranium and vanadium ores in Colorado before taking his present position, and it is probable that his methods will be adopted on a commercial scale in the new plants.

Bureau of Mines

The third annual report of the Director of the Bureau of Mines deals with the aims and objects of the Bureau, outlines its needs and gives a summary of work accomplished in its various departments. Some of the items are of special interest to the metallurgical industry.

During the fiscal year 1913 a study was made of milling methods in the lead-zinc districts of Missouri, Oklahoma, Arkansas, Wisconsin and Illinois. The investigation showed the possibility of practicable savings of ore now wasted. This work is to be continued and published in a bulletin.

The electric smelting of ores was under investigation at Pittsburgh. One problem studied was the possibility of using crude oil as a reducing agent, while another was the smelting of copper concentrates. In non-ferrous metallurgy the following points were considered: the possibility of recovering the iron in slag; of producing ferro-silicon from slags; of recovering sulphur from sulphide ores, and of discovering some suitable collector other than lead and copper for precious metals in smelting practice.

A study of sampling and assaying methods was under way during the year. This included the sampling of ores and metals, molten and solid alloys, charges in the furnace, and solid materials. It is hoped that the investigation will give methods that can be accepted as standard by producers and operators.

The work on smelter fumes and gases was conducted at San Francisco. This had special reference to the recovery of valuable constituents of fumes which are now wasted. New uses have been sought for sulphur, arsenic, bismuth, selenium and tellurium. Formal cooperation with the Bureau in advancing the work has been given by the Anaconda smelter in Montana and the Selby in California. A special report on the methods now in use or being tested for prevention of damage by smelter smoke is to be made by Mr. C. H. Fulton.

Other interesting studies have been under way in reduction of mineral waste, prevention of losses in making non-ferrous alloys, utilization of domestic fuller's earth, silica and minerals used in the making of refractories.

3. See this journal May, 1914, page 334.

Societies and Organizations

The Utah members of the American Institute of Mining Engineers have been organizing a local section of the parent body, preparatory to formulating plans for entertainment of the Institute at the meeting in Salt Lake City in August. Mr. R. C. Gemmel, manager of the Utah Copper Co., and Mr. Ernest Gayford, of the General Engineering Co., have been active in the work. The Utah engineers have asked the Nevada members of the Institute to join in the organization, which may be known as the Utah-Nevada section, provided the Nevada members acquiesce. The Salt Lake meeting of the Institute promises to be as successful as was the Butte meeting, and it is certain that Utah hospitality will make the affair most pleasant as well as profitable.

The International Engineering Congress has announced the topics under which the section on Mechanical Engineering will group the papers presented. Many of these topics are of direct interest to the metallurgical fraternity, as they deal with metal working, industrial management, power plants, etc. At present the Congress has received enrollments and subscriptions from 42 foreign countries. The subscriptions from the membership of the five American engineering societies, under whose auspices the Congress is held, is not so gratifying, and amounts to only 3.7 per cent of the total subscriptions. Procrastination on the part of American engineers is handicapping the work of the Congress, and they are urged to subscribe at an early date.

Flotation Litigation

In a recent circular to stockholders, the Ore Concentration Co., Ltd. (Elmore process), makes the following statement relative to its action for infringemen against the Sulphide Corporation, Ltd., in New South Wales.

"The judgment, which was delivered on the 6th of March, is very disappointing in one respect, inasmuch as the process as carried out by the defendants is held not to be an infringement of the patent sued under. On the other hand, it is satisfactory that a certificate of validity of this patent will be granted to us by the Court, and that all the alleged anticipations set up against it by the defendants have been swept aside. The position therefore at the present moment is that we have not proved infringement by the defendants in the above case, but we have established that our own patent is good and valid."

Anaconda-International Consolidation

Announcement is made that the Anaconda Copper Mining Co. has purchased the assets and business of the International Smelting & Refining Co. The International was organized several years ago, and its technical operations have been directed by men formerly in the employ of the Anaconda company. It has a lead and copper smelter at Tooele, Utah, and a refinery near Chicago. Anaconda company capitalists have been heavily interested in the International company, and the consolidation causes little surprise in view of the close arrangements formerly existing between the two companies.

Company Reports

Operations at the plant of the **Old Dominion Copper Mining & Smelting Co.**, Arizona, during 1913, were carried on under adverse conditions, due mainly to

congestion in the mining department caused by enlarging underground stations and cutting skip pockets. The construction of a new concentrator and the experiments that were carried on during the year made economical operation of the old mill a difficult matter. The new mill probably will not be in operation before August next.

Ore Extraction:	Dry Tons	Average Copper, Per Cent
Smelting ore	120,257	6.85
Concentrating ore	47,785	3.56
Silica lining	1,919	3.37
Total and average	169,961	5.88

Concentration:	Dry Tons	Per Cent Copper	Cost Per Ton
Company ore	47,972	3.56	.....
Custom ore	102,231	3.76	.....
Total and average	150,203	3.70	$0.878

Concentrating cost in 1912 was only $0.753 per ton. The increase in 1913 was due chiefly to abnormally heavy repairs in the mill, and reduced capacity caused by construction work. Experiments on flotation by the Minerals Separation process have been under way to determine the results that may be obtained by thus treating fine slime on which a heavy loss has been sustained in the past.

Smelting and Converting:

Tons new charge smelted	300,926
Cost per ton	$2.629

The cost was higher than in 1912 ($2.304) due to increased coke consumption, higher wages and extraordinary improvements. The smelting of flue dust was discontinued for three months, but was resumed when the furnaces showed no improvement in running without this material in the charge.

Pounds blister copper produced	30,811,441
Pounds fine copper produced	30,572,863
Cost converting per ton fine copper	$9.19

The old acid-lined converters have been replaced by the Great Falls type of basic-lined converter, and the results are satisfactory. The cost of converting in 1912 was $9.45 per ton of fine copper. The saving effected in 1913 was derived mainly from the decreased cost of relining, which fell from $2.23 per ton of copper in 1912 to $0.25 in 1913. The cost of improvements amounting to $24,732 is included in the cost for 1913. The total cost of producing fine copper was 9.70 cents per pound.

The annual report of the **Miami Copper Co.**, Arizona, for 1913, contains the following items of metallurgical interest. Experimental work conducted in one of the six sections of the mill has shown that the plant can be remodelled to handle at least 4000 tons of ore per day. Various mill improvements resulted in a lower cost of treatment, which is given below:

	Per Ton
Crushing	$0.20806
Concentrating	0.18852
Handling tailing and concentrates	0.05880
Water for concentrating	0.08553
Betterments	0.03134
Total	$0.57225

The results of milling operations for the year were as follows:

Ore milled	1,058,784 tons, 2.30% copper
Concentrates produced	45,410 tons, 38.09% copper
Copper in concentrates	34,597,568 pounds
Copper per ton ore	32.68 pounds
Mill extraction	71.06%

The cost of refined copper in concentrates on board cars at Miami, based on net smelter returns, was 7.9 cents per pound.

The company has developed about 6,000,000 tons of mixed oxide and sulphide ore, for which it hopes to develop a combination concentration and leaching process.

The 24th annual report of the **Wallaroo and Moonta Mining & Smelting Co., Ltd.**, South Australia, covers the operations during the year 1913. The report of the general manager, Mr. H. Lipson Hancock, states that the results of the flotation process at the concentrators has justified the expenditure made therefor, and is enabling the company to make a higher recovery of copper. The treatment of the slime tailing dumps at the Moonta mines, by leaching and precipitating, has been improved through a system of regular plowing which expedites oxidation of the copper minerals and yields a liquor richer in copper. Similar work at the Wallaroo mines has been discontinued in favor of flotation. One blast furnace in operation for 308½ days smelted 90,190 tons of charge. The output for the year was 7112 tons refined copper, 2161 ounces fine gold, and 1000 ounces silver 996 fine. The acid works produced 5602 tons sulphuric acid.

The Iron and Steel Market

While the stagnation in the iron and steel market was not greatly accentuated in May, there being, indeed, very little margin left for such a change, the dissatisfaction of the trade with existing conditions was greatly increased, because it became high time for an improvement to set in. The market lost ground constantly throughout 1913, and towards the close of the year a period of inaction lasting several months was indicated. Somewhat to the surprise of the trade a rather substantial revival occurred in January, and the recuperative power thus shown gave the trade much encouragement, so that it expected, after perhaps a lull of a month or two, a prolonged and marked improvement, particularly as the advent of spring normally brings increased consumptive requirements. As week after week has passed with the market showing somewhat increased stagnation, instead of marked improvement, the trade has become much depressed.

The output of finished steel in May has not been more than about 55 per cent. capacity, against about 65 per cent. in April, the highest rate this year having been close to 75 per cent., early in March.

Steel bookings during May have probably averaged somewhat less than 50 per cent. of capacity, and were at approximately a stationary rate during the month.

As in April, the course of steel prices has been downward, though not markedly so. Prices were already so near the cost of production, taking cognizance of depreciation and fixed charges and the increase in cost due to light operation, that all declines have met with stubborn resistance.

Towards the close of May a change occurred in sentiment, the trade becoming rather hopeful that some slight improvement at least would occur before midsummer, on the theory that stocks were altogether depleted throughout the country, with actual ultimate consumption at a rate in excess of the current mill shipments. Hopes for good business in the second half

of the year were augmented by the extremely favorable crop outlook, reinforced by evidences that the railroads were placing themselves in better position by the practice of severe economy.

Wage reductions, seriously discussed by independent mills in April as a probable necessity should conditions not improve within a short time, are still under consideration, but the positive attitude the United States Steel Corporation is known to have assumed against any wage reductions is likely to act as a deterrent influence, the independents fearing that they would be placed in an embarrassing position.

Pig Iron

May saw a decided revival in interest in the pig iron market. Several steel interests purchased basic pig iron for third quarter, and in one or two instances for the entire half year, while there was a marked increase in sales of foundry iron in small lots. In some quarters the increased activity of pig iron has been held to presage a general improvement in the iron and steel industry, but it will do no harm to reflect upon the fact that last summer witnessed a decided improvement in the pig iron market without any improvement in the steel market. What occurred then was that pig iron buyers put off their purchases until many were forced into the market at the same time, and, while their purchases advanced prices slightly, the market began to weaken again as soon as the buying was over, and eventually reached a lower level than ever. About the beginning of May it was quietly announced that Lake Superior ores would be sold for 1914 delivery at the 1912 schedule, involving reductions from 1913 of 55 cents on non-Bessemer and 65 cents on Bessemer ores, the announced prices being: Old range Bessemer, $3.75; old range non-Bessemer, $3.05; Mesabi Bessemer, $3.50; Mesabi non-Bessemer, $2.85, on Lake Erie docks. It is since rumored that sales have been made at 15 to 20 cents under the announced prices, but irrespective of prices the sales have not been large, and the "opening" even though delayed has been quite different from openings in most previous years, when large tonnages were placed under contract in a few days. The decline in ore from 1913 to 1914 is held not to necessitate or permit any decline in pig iron, it being maintained that the pig iron market in its declines had already discounted the reduction in ore, which was fully expected even before the close of 1913. Connellsville coke has weakened somewhat, there being a few contracts made for delivery after July 1 at well under $2, the price to which the majority of operators have steadfastly adhered for months. The pig iron market stands quotable as follows: No. 2 foundry, f.o.b. Birmingham, $10.50; delivered Philadelphia, $14.75 to $15; f.o.b. furnace, Buffalo, $13; delivered Cleveland, $14.25; f.o.b. furnace, Chicago, $14; at valley furnaces (90 cents higher delivered Pittsburgh), Bessemer, $14; basic, $13; No. 2 foundry and malleable, $13; forge, $12.50 to $12.75.

Steel

The market for billets, sheet bars and rods was quiet throughout the month, there being no prompt requirements to be filled and very little interest in deliveries after July 1, to which date consumers seemed to be well covered. Market prices are largely nominal, as follows: Billets, $20; sheet bars, $21; rods, $26, at maker's mill, Pittsburgh or Youngstown.

Finished Steel

Slight weakening occurred during the month in the majority of products. Prices named below are f.o.b Pittsburgh unless otherwise noted:

Rails, standard sections, 1.25c. for Bessemer and 1.34c. for open-hearth, f.o.b. mill, except Colorado.

Plates, tank quality, 1.12½c.

Steel bars, 1.12½c. base.

Shapes, 1.15c.

Iron bars, 1.30c., Pittsburgh; 1.22½c., Philadelphia; 1.10c., Chicago.

Wire nails, $1.55 to $1.60, base; plain wire, 1.35c. to 1.40c.

Sheets, blue annealed, 10 gage, 1.35c.; black, 28 gage, 1.85c.; galvanized, 28 gage, 2.80c.; painted corrugated, 28 gage, 2.05c.; galvanized corrugated, 28 gage, 2.85c.

Merchant steel pipe, ¾ to 3 in., 80 per cent. off list.

Steel boiler tubes, 3½ to 4½ in., 72 per cent. off list.

Standard railroad spikes, 1.40c., Pittsburgh, 1.50c., Chicago.

Structural rivets, 1.60c.; boiler rivets, 1.70c.

Cold rolled shafting, 66 per cent. off list.

American Electrochemical Society

The twenty-sixth general meeting of the American Electrochemical Society will be held at Niagara Falls on Thursday, Friday and Saturday, Oct. 1, 2 and 3, 1914. A local committee has been formed with Mr. L. E. Saunders as chairman.

Mr. F. A. Lidbury, works manager of the Oldbury Electrochemical Company, of Niagara Falls, is the president of the society. Dr. Joseph W. Richards, Lehigh University, Bethlehem, Pa., is the secretary.

BRITISH IRON AND STEEL INSTITUTE

The annual meeting of the Iron and Steel Institute was held in London on May 7th and 8th, 1914. Annual reports of officers were presented and the president-elect, Mr. Adolphe Greiner, was introduced into office. The Bessemer gold medal for 1914 was presented to Mr. Edward Riley. Following is the list of papers on the program:

The Forms in Which Sulphides May Exist in Steel Ingots. By J. O. Arnold and G. R. Bolsover.

Experiments on Allotropy of Iron (Behaviour of Ferro-Magnetic Mixtures; Dilatation of Pure Iron). By C. Benedicks.

Recent Developments of the Iron and Steel Industry in India. By A. Bose.

The Recrystallization of Deformed Iron. By C. Chappell.

The Hardening of Iron and Its Alloys. By C. A. Edwards and H. C. H. Carpenter.

Influence of Molybdenum Upon the Corrodibility of Steel. By J. N. Friend and C. W. Marshall.

Note on a Curious Case of Decarburization During the Hardening of Steel Dies. By H. C. Greenwood.

The Magnetic and Mechanical Properties of Manganese Steels. By Sir Robert A. Hadfield and B. Hopkinson.

Case-Hardening. By H. L. Heathcote.

Failures of Heavy Boiler Shell Plates. By S. A. Houghton.

Production of Steel Direct from the Ore. By E. Humbert and A. Hethey.

Theory of Hardening. By A. McCance.

The Colorimetric Estimation of Sulphur in Pig Iron and Steels by Paper Impregnated with a Solution of Arsenious Anhydride in Hydrochloric Acid.

The Development of Dry Cleaning in Blast-Furnace Gas Purification. By F. Muller.

A New Reagent for Etching Mild Steel. By W. Rosenhain and J. L. Haughton.

Talbot Process at Witkowitz. By F. Schuster.

Is There a Complex-Ore Problem?

A Symposium on the Economics and Metallurgy of Mixed Base-Metal Sulphide Ores, Showing Present Status and Future Prospects of This Branch of Ore Treatment

In our May issue we published the first batch of replies received from a number of prominent metallurgists to an inquiry submitted to them in form of the following six questions:

1. Is there a "complex-ore problem," and what, in your opinion, constitutes a "complex" ore?
2. Would such ores be amenable to present methods in the hands of skilled operators, or is there a need for new methods of treatment?
3. Do you believe there are many mines now idle which could produce a considerable tonnage of such ore if suitable methods of treatment were known?
4. What element or quality of such ores makes them refractory to present processes?
5. What methods or processes do you regard as probable solutions of the problem (mechanical, hydrometallurgical, electrometallurgical)?
6. To what extent would microscopical examination of such ores be of material assistance in suggesting proper methods of treatment?

The following replies, since received, will be found equally interesting:

Stuart Croasdale.—If we are to adhere to the usual definition of an ore, as a mineral or group of minerals which can be worked at a profit, many of the so-called complex-ore bodies do not exist. If we are to consider as complex-ore, what one engineer has aptly termed a "mineral aggregate," or a group of minerals which cannot be worked at a profit, individually or collectively, by existing methods or under existing conditions, but which have an assumed value based on the gross market value of the merchantable metals contained therein, then the field might be broadened even to the sands of the sea.

However, placing both ore and "mineral aggregate" containing from two to five of the more common merchantable metals under the term of complex-ore, I think the available quantity of this material is greatly overestimated. Improved methods of ore treatment may increase the number of producing mines to some extent, but my experience has been, in examining properties of this character, that the ore bodies frequently are too small to justify the erection and operation of a reduction plant, and too low-grade to ship crude to a custom plant, except in very favored districts. When the "ore in sight" will not pay, or more than pay, for the property itself and a suitable metallurgical plant, the problem becomes very simple. On the other hand, we all know there is a proportionate number of good properties containing large tonnages of complex ores, which are now standing idle or are operating at an unsatisfactory profit.

The only method now in use for the treatment of these ores is the mechanical concentration and separation of the minerals into marketable products for lead and zinc smelteries. This is necessarily done at the mines to avoid transportation of large quantities of gangue or worthless material. Losses occur at every turn. A typical recovery by this method in a Colorado mill, equipped with every type of device for concentration and preparation of commercial products, is as follows, based on the assays of crude ore and tailings: Gold, including free gold sold as bullion, 75.9 per cent; silver, 50.6 per cent; lead, 86.7 per cent; zinc, 58.6 per cent; copper, 55.0 per cent. Higher recoveries have to be sacrificed in order to maintain the products at a marketable grade. To the above losses must be added smelting charges, deductions, penalties, and metals in each product not paid for, the amount of which is appalling.

As an illustration of these losses, a recently published statement relating to a Montana property is significant. The ore can scarcely be called complex, for it was milled for zinc alone. Concentration and flotation are said to recover over 90 per cent of the minerals in the crude ore.

The zinc product contained 50 per cent zinc, 2.4 per cent lead, 0.5 per cent copper, 26.3 oz. silver and 0.05 oz. gold per ton, and cost $4.56 per ton to produce. This concentrate had a market value at the mill of $23.79 per ton. The gross value of the merchantable metals in this product, based on current quotations, was $80 per ton. Therefore, $56.21 per ton disappeared between the mill and consumer. This paid legitimate freight and smelting charges, of course, but mostly it represented metals not paid for which lie in the "residue" dumps at the zinc smelteries. How much greater are similar losses from complex ores?

Surely the present method of treatment is not all that can be desired, and it is scarcely conceivable that the cost of treatment in a more direct and refined metallurgical process, whatever it may be, will exceed the cost, plus the losses sustained by mechanical concentration and smelting.

The refractory character of complex ores is usually due to the chemical combination or physical arrangement of the minerals, and a microscopical examination of such ores will undoubtedly aid in an intelligent solution of the problem of their treatment.

No method of treatment can be considered the best for all complex ores. Each ore is a problem in itself. In some ores the minerals are coarsely crystalline and easily separated into comparatively clean high-grade products. Such ores, even, with the attendant losses, may be best adapted to mechanical concentration.

Other ores, when examined under the microscope, will reveal the minerals intimately associated in very small crystals, or perhaps in non-crystalline form, which, if separated, will involve so much loss in sliming and subsequent smelting that any form of mechanical separation would be condemned as impracticable if other processes were available.

In still other ores chemical combination exists in the form of double sulphides of the metals. The rational treatment of such ores certainly is not mechanical beyond the removal of a portion of the gangue, for no product can be made that will not involve a considerable loss in the form of a metal not paid for.

As stated before, only one method of treatment now exists—that of mechanical concentration and separation. The consulting engineer cannot recommend the installation of untried processes, no matter what his opinions and experimental results may be. His clients are like the gentleman from Ohio in the selection of horseflesh. They do not want a process that "has been" or "is to be." They want an "izzer." Therefore, the engineer wears the ore to a frazzle sending it through the interminable mechanical grooming and gets the best results he can with from 20 per cent to 30 per cent combined loss in tailings; his lead product containing 10 per cent unrecoverable zinc, his zinc product containing 10 per cent unrecoverable lead and more or less unrecoverable silver—all of which is, of course, unpaid for, if not penalized. If he can figure a profit when he is through, the system is installed; if not, the property

stands idle awaiting a better method of treatment.

New processes cannot be put into operation by the discoverers nor by stock companies. They must be matured by companies that are working complex ores at a profit under existing methods and wish to improve their condition. This can be readily done by appropriations set aside from the earnings, which would virtually come from the properties themselves and not from the shareholders. In case of failure the loss of such an investment will not be felt, and in event of success the science of metallurgy, as well as the mine, will be amply rewarded.

If managing engineers in charge of properties of this kind would devote more time and energy to the development of new processes along lines best adapted to their ores, whether that process be hydrometallurgical, electrometallurgical or pyrometallurgical, I believe we would get out of the mechanical rut and make considerable advance in the treatment of complex ores.

There is a decided tendency in this direction at the present time among the large copper companies, particularly in connection with the treatment of low-grade porphyry ores, where the concentration and smelting losses are between 35 and 40 per cent, and also where large quantities of oxidized ores exist for which there has been no method of commercial treatment available. It is a pleasure to hear of the extension of this work to lead properties, and it is to be hoped that the complex ore properties will soon fall in line.

Much could be gained by combined and organized research among these respective types of properties, whereby a great deal of duplication of work and discouragement could be avoided. I know of a number of cases where work of this kind has been duplicated independently by engineers, and abandoned at the same point; whereas, if the second engineer could have profited by the experience of the first, both would have gained in coin and reputation, and a positive advance would have been made in metallurgy. If it is impossible, as some have said, to overcome professional jealousy in organized research of this kind, why not permit the Bureau of Mines to co-operate with and receive financial assistance from mining companies for such work in proportion to the benefits obtained, and thereby develop an "all star," impartial, national research staff.

Denver, Colo.

H. M. Parks.—I have noted, in being around different mining camps, the general opinion which your questions suggest. Although the miner and prospector often speaks of the question as "the complex-ore problem," it seems to me that it might more properly be designated a phase of a "complex-ore problem." In other words the mixture of base-metal sulphides is an ore amenable to treatment by existing processes at a higher cost and treatment loss than one carrying only one valuable metal. Accordingly such a mixture, to be an ore in the present accepted meaning of the term, must have a total value considerably in excess of that needed when the value is all represented in one metal.

There certainly are millions of tons of rock containing metal values of such grade that they cannot now be treated at a profit, which may sometimes become profitable through the invention and improvement of processes, improvement of transportation facilities, etc. Many of our "complex ores" coming into this category have total metal values greater than other ores in the same locality which are being treated profitably for only one metal. That ore A worth $5 is being treated profitably, and ore B at the same mine worth $8 cannot be treated at profit, causes much comment; but when we consider that the valuable metal in A has but to be separated from the gangue, while several metals in B must also be separated from each other, we see that it is about as imperative to demand a new process to handle ore A as ore B.

We do need and probably always shall need improvement in our processes for treating ores, whether simple or complex; but it is probably true that the field is larger for treating the complex ores because the problem is more variable. The current impression that the "complex ore" is a present day metallurgical bug-a-boo is quite erroneous.

Corvallis, Ore.

Robert H. Richards.—Answering your questions briefly, will say that:

1.—A complex ore is one that is fine-grained, containing minerals that are difficult to separate, but which would have greatly increased value if separated.

2.—Some of these ores would yield to present processes skilfully applied, but others need new processes for their profitable treatment.

3.—It is reported that there are many mines now idle which could produce a considerable tonnage of ore if suitable treatment processes were known, but I am not in a position to know.

4.—Zinc seems to be the worst stumbling block in the metallurgy of complex ores.

5.—In my opinion all three of the types of processes mentioned—mechanical, hydrometallurgical, electrometallurgical—either separately or collectively, will be used in solving the problem.

6.—The use of the microscope would aid in determining the minerals present.

Boston, Mass.

W. McA. Johnson.—Metallurgically speaking an ore may be defined as mineralized material containing metal in sufficient amount and purity as to permit its profitable extraction. The terms "simple" and "complex" are relative and can best be illustrated by examples of each class. For instance, a simple ore is one that contains one metal constituent only, or so great a proportion of one metal as to permit of its extraction by a simple furnace treatment, the subordinate metal or metals, if present, not interfering with the extraction of the chief metal. This is exemplified by the ordinary sulphide or oxide ores of lead, of copper, or of zinc. When, however, the ore contains two or more metallic minerals in such a physical form as not to admit of separation by mechanical methods, and if the contained metals could not be easily extracted by present furnace methods, such an ore would be called a "complex ore." This might be exemplified by an intimate mixture of galena, blende and chalcopyrite or chalcocite—say 30 per cent zinc, 10 per cent lead, with or without a small percentage of copper; or copper 3 per cent and zinc 30 per cent. In cases of this kind the percentage of zinc present is too great to admit of smelting economically in the lead furnace, and the percentage of lead too great to permit of the profitable extraction of zinc in the ordinary retort furnace.

The question of the proper method of treating complex ores, such as those just quoted, constitutes the problem, and it is one that has remained unsolved in many instances.

Your second question, as to the amenability of complex ores to present day processes, provided skilled operators conducted the treatment, practically answers itself in the fact that there still remains the problem. Metallurgy is always progressing, and that which was good practice a few years ago is now considered bad practice; losses in slags and tailings permitted in the state of the art ten years ago would not now be allowed. For example, the porphyry copper ores of Utah could not have been treated profitably by proc-

esses of a decade ago, yet today they yield handsome returns—and even now the present practice is far from perfect, in that the tailings from the mechanical concentrators are stored up for future extraction by methods not yet devised or perfected. Hence an ore may be refractory today and easy to treat by improved or new processes tomorrow. The application of the laws of physical chemistry and commercial science, cemented by common sense and due diligence, will solve many a problem which heretofore has been too complex for a profitable solution. Eternal experimenting is the price of progress.

I decidedly do think that there is a need for new methods of treatment of the so-called complex zinciferous lead or copper ores, and I have devoted ten years of careful study and research to develop my present process, which I believe promises to settle many perplexing questions of metallurgical treatment.

As to possible idle mines which could produce a considerable tonnage of complex ores if suitable methods of treatment were available, I will say that I recently conducted a two-months' advertising campaign which had for its object the gaining of information as to the need of a process which would extract practically all of the metal content of zinciferous lead or copper (or both) ores, and my files are filled with letters from all parts of the country. In most of these letters the ore mentioned is not now being marketed, although the total combined metal values show a good working margin. Whether these deposits represent the "millions of tons" referred to in your letter I have not as yet computed, but they appear to be quite extensive and in need of some process which will render their treatment commercially profitable.

The Johnson electric furnace process, as developed to its present state of efficiency, can extract from ores practically all of the contained zinc, lead, copper and precious metals, in one smelting operation, thus avoiding the wasteful process of concentration in which there is an unavoidable and large loss of mineral in the tailings, and eliminating the subsequent heavy penalty charge for treatment by the smelter of the large percentage of zinc usually present. Moreover if one considers that during the past year the value of the zinc which was thrown away in the slags of American smelting furnaces aggregated $30,000,000 (equivalent to two-thirds the output of zinc in the United States for that year), the great importance of a process which will **save the zinc** in these complex ores is manifest. This question is answered in the strong affirmative by the contents of the many letters received.

In reply to question 4, I should say that the cause of the refractoriness of these complex ores is chiefly the presence of a high percentage of zinc, the ores being so constituted physically as not to be amenable to separation of the component minerals by mechanical treatment.

I consider that the Johnson electric furnace process will solve most of the problems now under consideration; and while there may be future progress in the mechanical and hydrometallurgical fields, their application for the treatment of zinc ores does not seem to me to be promising of satisfactory results.

Your final question as to whether or not a microscopical examination of complex ores would be of material assistance in suggesting proper methods of treatment, is one of interest; but the application would be of value chiefly as an aid to determine the proper methods of mechanical concentration. If the ore were rich enough it could be smelted direct in the electric furnace, but a case might arise in which it would be desirable to concentrate the ore (even though some of the mineral content be lost) prior to smelting. As a general rule the details of the electric furnace treatment have to be determined for each individual ore, if wholly satisfactory results are to be attained.

Hartford, Conn.

Francis A. Thomson.—1.—There is, I think, in a limited sense a "complex-ore problem," but the problem is less serious and not as general as is commonly supposed.

By a "complex ore" I mean an ore containing sulphides of copper, zinc and lead in sufficient quantities to justify the recovery of all three metals, with of course the usual accompaniment of gold and silver.

2.—Such ores are in part amenable to treatment by present processes when properly conducted. This is more true of the recovery of the copper and lead than of the zinc. There is a moderate demand for a process which will permit more complete recovery of all three metals than is now possible.

3.—To my knowledge there are very few mines lying idle because of the complexity of their ores. I have had occasion to examine many mines, which were claimed to be idle from this cause, but have never yet found the claim to be true. Quantity or value of ore was lacking in every case.

It is my experience that this claim is often the means by which ignorant or deceitful persons impose "fake processes" and fraudulent promotions upon the public.

4.—There are mainly three causes of the refractoriness of complex ores:

a. The intimate intermixture of the sulphides mentioned.
b. The lack of difference in specific gravity between chalcopyrite and sphalerite.
c. The volatility and ready oxidability of zinc.

5.—For (a) and (b) in the preceding classification, magnetic or electrostatic separation seems to promise best. Flotation also may come to have an important place.

For (c) electric smelting would seem to have possibilities in view of the ease of control of furnace atmosphere and temperature. Of course, proposals for a sulphatizing roast of the Ziervogel type, with subsequent leaching with water, or an oxidizing roast with a sulphuric acid leach, the acid being made from the gases of the roasting operation, are always with us and may some day displace igneous concentration, but this is doubtful.

In general the progress of the future will be like that of the past—slow improvements step by step rather than sudden changes of a revolutionary character.

6.—Microscopic examination of any ore as a preliminary to devising a treatment scheme is exceedingly valuable, and all of my students are being trained to appreciate its value and to make use of the microscope daily for problems of this kind.

Pullman, Wash.

Lewis B. Skinner.—I would say, in the first place, that what is generally termed a "complex ore" is a mixture of the various metal sulphides such as zinc, lead and iron. Such ores may or may not contain gold and silver, but usually do. There is usually some gangue with these sulphides, which is commonly of a silicious nature, but may be lime, dolomite or barytes.

In answer to the second part of the question, would say that there certainly is a complex-ore problem. There is also a gold-ore problem, a lead-smelting-ore problem and, in fact, problems in the handling of all ores. Every one who is trying to work as efficiently as possible is trying to cut down the manufacturing

costs of all of the metals. Since complex ores usually contain the metal zinc as the most valuable constituent, I refer more particularly to zinc itself when referring to complex ores.

We hear a great deal about the terrible loss that obtains in reducing metallic zinc. This is no worse than that obtaining in the production of other metals, i.e., putting them into merchantable shape. Were we without any processes at all for the manufacture of zinc or spelter, we would then have a real problem; but we are not without methods, which, while not as efficient as any one desires, are still able to compete against a given market and turn out a sufficient supply of metal.

2.—Usually the first step in the production of any of the metals is ore-dressing. The ores as mined are of too low a grade to be handled directly in any smelting operation. It is not a fact that poor methods make the losses high, but that the handling of ore-dressing methods is deplorable. There are only a few plants practicing ore-dressing that do so with the idea of getting every pound of available material. It is usually only a question of whether the mill, taking its own mine ore, is paying a profit. If it does not, the mine is blamed for it. If it does, all are satisfied. The poor quality of mill operation is no doubt responsible for the notion that there is a complex-ore problem. To offset this many experimenters have undertaken to get up a new mode of procedure which, in itself, usually proves to be much more cumbersome, if not apparently more wasteful, than the ones that are commonly used and yet condemned owing to lack of ability in the operator.

A great improvement could be had if the present methods of ore-dressing were handled in a business-like and satisfactory manner; but it would be very foolish to say that some new procedure would not prove, in some cases, more desirable than the known methods. You, Mr. Editor, and I know, that there is too much of a general tendency on the part of persons advocating new schemes to want to throw all of the good, known, tried processes out of commission and immediately start off on new lines.

It is possibly several decades since the use of platinum sponge as a catalyzer for the union of sulphur dioxide and oxygen was first announced. It has taken all of these decades with much expensive and strenuous investigative work, coupled with high engineering ability, to evolve schemes which are now known as "contact processes" for the manufacture of sulphuric acid. Even at that, the contact processes will compete only under given commercial conditions. When, therefore, a person drops into the complex-ore field and immediately announces a new process, one is inclined to be skeptical as to the engineering of it, at least, although the fundamental principles involved may seem to be sound. It has been repeatedly proven that a simple procedure, coupled with high operative efficiency, usually beats any intricate procedure having too great a demand for high class labor and care in watching the various steps. The answer in any case is a matter of dollars and cents. If the old procedures, which may seem to be somewhat wasteful, cost little for operation and something more for actual loss of metal at its valuation at the point of treatment, and these costs prove to be less than those of some more elaborate procedure having possibilities for higher recoveries, it is merely a question of the survival of the fittest, which, in this case, will be the scheme first outlined. With the possible exception of electric zinc smelting as now being tried out, it has been proven under any conditions that I know of that complex ores should be treated by simple ore-dressing, followed by the old procedures, as against any new chemical or other sort of a scheme outlined to date.

3.—I do not think there are many mines now idle for lack of suitable treatment processes. I know of several "scouts" of the large companies who complain that they are unable to locate such large tonnages of ore as several enthusiastic people are inclined to believe exist.

4.—There are two conditions which might obtain: one is where the various constituents are so finely disseminated that extremely fine grinding is necessary; and the other where the mineral particles are composed of the metals upon which separation is desired. In other words, chemical combination or fine mechanical combination makes these ores refractory to present processes.

5.—Mechanical methods, coupled with the various smelting processes for the different metals, is the general procedure now adopted. The hydrometallurgical schemes do not seem to offer much for the reason that they necessitate the handling of salts of high heats of formation in large volumes of solution, calling for a high investment cost and demanding expensive chemicals. I think the electrometallurgical presents the only line worth while, in that either the electrothermic or electrolytic methods may be undertaken. With regard to the latter, one must be sure of the commercial aspect and first go thoroughly into the power situation. With the prospect before us of cheaper power from hydroelectric stations, some electrometallurgical procedure should be evolved which will ultimately yield cheaper metal.

In passing, since there are fads in metallurgical lines as well as in others, I might note that we are going through a stage when the whole fraternity is on the qui vive trying to make every ore amenable to treatment by some flotation process. In the case of one-mineral separation in the United States this process has proven an aid; but it is of no benefit in the case of ores of zinc, which are truly complex, although being advocated for such. I do not know of any complex ore in Colorado, for instance, that could possibly show any advantage under this treatment.

6.—Almost every one examines ore with a magnifying glass or even goes so far as to use a microscope. We use both and are guided thereby. Almost every one in the ore-dressing business or ore-handling business uses on or the other and gains a great deal of information. When it is indicated by magnifying that extremely fine crushing is a necessity, mechanical separation then becomes out of the question, unless under peculiar circumstances.

Denver, Colo.

Arthur J. Hoskin.—1.—Yes, such a problem does not exist. To me a complex ore is one that contains or is made up of two or more base-metal minerals—not necessarily sulphides—each of which could be marketed if suitably separated or otherwise prepared to fit commercial demands. In some instances of ores containing more than two minerals, it may be sufficient to make but two products. Thus, if an ore contain galena, sphalerite and chalcocite, the three metals thus represented could each be realized upon in the market if the sphalerite were removed cleanly from the other two minerals. I once considered an ore composed principally of sphalerite and barite complex. I had such an ore and was assured of a good market among paint manufacturers for clean oxide of zinc and sulphate of barium. It was before the days of electrostatic, flotation and magnetic separations. The minerals of the ore could not be effectively separated by the then existing table methods, either wet or dry, as their specific

gravities were nearly the same. To me it was complex, but both minerals were not sulphides. I would certainly call a mixture of molybdenite and biotite complex, since I understand that these minerals do not separate well under any of the present processes, and yet there is a market for each mineral when clean.

2.—Certain combinations of minerals that are now classed as complex ores are undoubtedly amenable to present methods in the hands of skilled operators; but there are other ores, the satisfactory dressing or treatment of which has not been discovered.

3.—I know of a few mines lying idle from this last cause, and I presume there are many.

4.—Among the physical properties that have thus far proved obstacles in the thorough separation of such minerals I would mention similarity in specific gravity; shapes of crystals; similar action in flotation; equivalent electrical properties; similar magnetic properties; equal solvency in chemicals; similar action under heat.

5.—It would seem that there is a field for all three general sorts of treatment.

6.—The microscope would reveal such peculiarities of the physical structures of the associated minerals as would be useful in deciding upon the degree of fineness to which ore should be ground, and it might also determine the most desirable method of pulverization.

Denver, Colo.

George H. Scibird.—I define a complex ore as a difficult ore, or one which is refractory to profitable treatment by ordinary prevailing processes; and with this definition in mind, the statement that there has existed a complex-ore problem, locally, becomes true. To convey some idea of the nature of one such ore, the following analysis is given:

Gold	0.11	oz. per ton
Silver	11.00	" " "
Insoluble	85.00	per cent.
Iron	2.75	" "
Manganese	2.25	" "
Lime	3.00	" "
Copper	0.60	" "
Lead	1.00	" "
Zinc	1.25	" "

The character of this ore is such that concentration by any practicable method was unsuccessful, while only fair results were ever obtained by combination of various processes.

In this particular case, it is improbable that skill on the part of the operator would have helped the situation, because the metals existed in the ore in such a condition that their recovery apparently was impossible within commercial bounds. In this respect the ore was not amenable to treatment by existing, proved processes. Thus it became evident that either a new process must be found, or some old process, with modern methods applied, be revived. In fact, this has been done. In developing the process an effort was made to overcome the weaknesses of the old chloridizing-roasting process, such as loss by volatilization, high cost and low extraction, and by improved chemistry in the use of solvents. The application of modern, highly specialized machinery will also prove of great benefit in the new plan of treatment. Thus, the Christensen process, strictly speaking, is not new when considered as a revival of the chloridizing process, but it embodies new ideas in a very large measure, and is so improved that it bids fair to take a most important position in the treatment of what have been complex ores in the sense of my definition.

While there are not, locally at least, many mines that are actually idle for want of such a process, there are many which are seriously hampered by a vast tonnage of non-smelting-grade ore which may well become a valuable asset with the adoption of this treatment. When, however, the country at large is considered, there are several properties, now idle, which can readily be remembered, that could be made profitable by the new process, and there are more which might operate to better advantage with this method of treatment than under their present process.

There are several features which have proved detrimental in connection with the ore. First may be mentioned the value. This figure is such that smelting is out of the question. Secondly, the base metals occurring as carbonates and chlorides, with the gold in an extremely fine state of division, makes concentration unsatisfactory; while cyanidation either alone or in combination with concentration is only fairly successful. Leaching with various solvents, roasting, and in fact the whole category of usual procedures, is not promising. The new process, however, promises to save the situation.

Thus, at least locally, and to a considerable extent, generally, the modernized chloridizing-roasting and leaching process will prove of immense value in solving the difficulty, although where the gangue is limestone this process is not perfect. It will, however, make available the value in a tremendous tonnage of ore which is at present worthless.

Salt Lake City, Utah.

R. D. George.—My various trips to the San Juan and other districts in Colorado, particularly in Boulder county, have led me to the opinion that there are large quantities of low-grade ore in the State which cannot be profitably treated by the present methods. A great deal of money is spent on processes, plants, and mills of various kinds without scientific investigation of the material to be treated, and the adaptation of the methods to local conditions.

1.—I have heard the term "complex ore" applied in the following cases:

a. Where the metals contained in the ore are in minerals of like specific gravity and therefore difficult to concentrate. An example of the kind would be zinc sulphide and pyrite.

b. Where the metallic contents are so disseminated through the gangue material as to make concentration difficult, if not impossible. This is the case with some of the tungsten ores of the eastern part of the Nederland area where cherty quartz and ferberite are so intimately mixed in such small grains that any type of crushing leaves the ferberite grains with sufficient gangue attached to make a low grade concentrate, and the quartz grains carry away sufficient attached ferberite to make the percentage of saving low.

c. Where there is considerable variation from place to place in the quality of the metallic mineral contained, as in the San Juan region where the silver, for example, may occur in any one of ten different forms, such as native silver, argentite, cerargyrite, stephanite, pyrargyrite, freibergite, proustite, silver-bearing tetrahedrite, silver-bearing tennantite, and argentiferous galena, along with pyrite, chalcopyrite, sphalerite, anglesite, hinsdalite, and a dozen others, and gangue minerals such as rhodochrosite, rhodonite, jasperoid, etc.

d. Ore deposits in which no definite line can be drawn between the oxidized and the unoxidized portions and in which sulphides, carbonates, oxides, and tellurides are intimately mixed.

e. To deposits in which many metallic bases of

value are mingled in the same ore, as in some places in the San Juan region where operators are puzzled to know whether their values are chiefly in the form of lead, silver, gold, copper (tungsten), or zinc, or all of these together.

2.—I believe that modifications of present processes to meet local needs could be made by experienced and skillful operators in such a way as to overcome many of the difficulties. Again cooperation between treatment plants would make it possible for specialization in mills to meet the demands of certain types of ores. In the past, the custom in building mills has been to generalize in such a way as to meet every possible type of ore. This seems to me to be contrary to scientific principles and economic considerations.

3.—I am more familiar with conditions in Boulder county than with those in most other parts of the State, and I am convinced that, with suitable concentration plants and other treatment processes, idle mines of Boulder county could furnish a very large tonnage. This is also true of parts of Gilpin and Clear Creek counties, particularly the Georgetown and Silver Plume areas.

4.—This question has been answered rather completely under No. 1.

5.—I believe that the varieties of ore and the purposes for which they are treated are such as to demand mechanical improvements in concentration, modifications of the hydrometallurgical processes, and to warrant more thorough and careful electrometallurgical experiments.

6.—I have long advocated the careful microscopic examination of the so-called complex ores as an aid to the mill man and the metallurgist. I talked this matter over with Dr. Holmes of the United States Bureau of Mines some years ago, and urged the desirability of attacking the problem from this standpoint. I also outlined a method of attack, and the data to be sought in such methods. I believe that some work has been done since by the Bureau of Mines, though I do not know the extent of the work or its results.

Boulder, Colo.

S. E. Bretherton.—Yes, there is a complex-ore problem. One type of complex ore is a sulphide containing both zinc and copper, zinc and lead, or zinc, copper and lead, with or without silver and gold. I might name another complex ore, low-grade, silicious, oxidized copper ore. The absence of sulphur and high percentage of silica prohibits such ore being smelted for the production of copper matte, if such ore is located in districts where iron sulphides are not plentiful. Concentration of such oxidized copper ore is not practical, and the presence of alumina, iron and often some lime makes leaching methods by the use of acid too expensive, even if cheap metallic iron should be available for the precipitation of the copper after it is obtained in solution. The leaching of either copper or zinc ores when oxidized, often called carbonates, is not generally suitable for the ammonia carbon-dioxid process, as such oxidized ores are usually mixed with considerable quantities of silicates of copper or zinc, which prevents a high extraction by this last named process. For this reason I am more particularly interested in the treatment of sulphide ores, from which, after an oxidizing roast, the zinc can be recovered by leaching with ammonia and carbon-dioxid, and the other valuable metals recovered by the usual methods of smelting.[1]

[1]See "Preparation of ore containing zinc for the recovery of other metals, such as silver, gold, copper and lead, by the elimination and subsequent recovery of the zinc as a chemically pure zinc product," page 1481 in the August, 1913, *Bulletin* of the American Institute of Mining Engineers; and F. L. Wilson's article on the same subject in the *Mining & Scientific Press* of March 14th last, page 453.

2.—When the ore is a sulphide and of suitable crystalline structure, zinc and lead can generally be separated by some of the ordinary methods of concentration, and sometimes zinc and copper can also be separated, but usually with excessive loss of gold and silver when the ore is rich in precious metals; but in a great many cases the structure of the ore will not permit of this mechanical separation. I believe there is a need for new treatment processes.

3.—Yes, if zinc could be added to the value of the ore in addition to the other metals, instead of being deducted by being penalized when the other metals are sold to the smelters as at present.

4.—Zinc is difficult to separate from copper by concentration. If the ore is smelted direct, the cost to recover the copper is increased by the presence of zinc, and the presence of zinc is much more objectionable when smelting an ore for the recovery of lead. (See answer to Question No. 1.)

5.—The processes most suitable would depend very much on the location of the mine where the ore is to be treated, as well as the analysis and structure of the ore. In some cases for sulphide ores, all three of the processes named may be required. Electric smelting with very cheap water power may come to the assistance of the retort or fire process, but even then considerable fuel is needed as a reducing agent with the roasted zinc ore, and a very expensive plant installation. For this reason it is reasonable to expect more development for the treatment of complex sulphide ores along the lines of concentration and leaching, and leaching direct. The complex zinc and copper sulphide ores of California are different in their structure to ores of the same analysis in Colorado, Utah and Arizona, and no known method of mechanical concentration is suitable for the ores of Shasta Co., Cal. Containing, as they do, considerable alumina and lime, any acid process does not seem suitable. For this reason the writer has developed the ammonia carbon-dioxid leaching process, which, so far, seems the only suitable process, requiring no fuel excepting oil which is the cheapest fuel in California.

6.—I would consider microscopical examination of any complex sulphide ore of material assistance in suggesting proper methods of treatment, as no doubt by experience the metallurgical engineer could decide in a short time whether mechanical concentration was suitable for the ore to be treated or not. We find that a powerful magnifying glass is a great assistance to us to quickly decide when the ore is properly roasted, and a more convenient and quicker means for the man in charge of the roaster than is the ordinary laboratory test.

San Francisco, Cal.

William G. Haldane.—1.—I think there is a "complex ore problem," although the term is used rather indiscriminately. I apply the name "complex ore" to such as exists in the San Juan region of this state, finely disseminated and intimate mechanical mixtures of the sulphides of copper, iron, lead, and zinc, and others of a similar nature.

2.—I believe that present processes in combination in the hands of skilled operators would accomplish much towards the solution of these problems, due to improvements and developments in the processes themselves, resulting from experimentation and use by such operators.

3.—There is not "millions of tons" of such ore, but an attractive field would be opened up if new processes were developed.

4.—The difficulties encountered in processes at present are largely mechanical, and result also from the

attempt to use the same medicine for a number of widely different ills.

5 and 6. Am greatly in favor of combination methods. I think the tendency has been to apply to an ore containing a number of minerals, each with its own special characteristics, a process involving only one principle, as, for example, specific gravity. To illustrate further, an ore may contain, let us say, minerals 1, 2, 3 and 4, with 1 and 2 of high sp. gr. and 3 and 4 of low sp. gr. In this case a preliminary concentration and separation may be effected with tables. Minerals 1 and 2 may be widely different magnetically, or may be readily separated with electrostatic machines, and so on.

I thoroughly believe a microscopic examination would be of very great assistance as a preliminary step in the examination of these complex ores. It reveals properties and information at first hand that should be invaluable to an investigator. The microscope should be one of the first pieces of apparatus resorted to by every technical man in the investigation of the characteristics and physical features of an ore.

I am firmly of the opinion that our problem is a metallurgical one, but I believe closer attention to details and manipulation of our present processes will result in higher efficiencies and greater economies.

Golden, Colo.

Warren F. Bleecker.—1.—The term "complex ore' is an unfortunate one, because it does not mean anything specifically, but may be applied to any ore containing two or more valuable metals. In certain Colorado mining districts when one speaks of a complex ore he refers especially to base-metal sulphides containing zinc. In this sense the term is more or less reflective, since the method of treatment is the thing which is really complex. Unless one had the authority to limit the meaning of this term, he certainly could not embrace all of the meanings in a definition of unlimited length. For example, the mineralogist might place three vanadium minerals in the following order of complexity: roscoelite, carnotite, patronite. The technologist would certainly reverse the order. And I have reasen to think that the analytical chemist would establish still another order of complexity for these three common vanadium-bearing minerals.

It is apparent, however, that the question refers to the base-metal sulphides. Those of the base-metal sulphides may be called complex which cannot be treated by standard mechanical devices; or in which no one valuable metal is in sufficient quantity to warrant profitable treatment; or in which there is a small quantity of zinc or a larger quantity of zinc which is chemically combined with the other values. It is obvious that if there were some cheap chemical which could be applied to cause the metals to be quickly and cheaply separated, the term complex would cease to be applied.

2.—Each ore presents a separate problem, be it a complex ore or a placer dirt, and the only method of getting at the truth will be the examination and report made by a reliable laboratory; or, still better, to send several carloads to a mill operating on a similar ore. If the mine is not exhausted by the loss of this quantity of ore, and the milling process is even moderately successful, the proposition would be very hopeful.

3.—I do not believe that there is a large quantity of good ore now developed and awaiting suitable methods of treatment. At the same time the present methods of ore-treatment are deficient, and development is held back accordingly. As known methods are improved and new ones devised, there will still be ores which are just on the line, and which will make further demands on the technologist.

4.—The quality of complex ores which contributes chiefly to their refractoriness to present processes is the distribution of values among several elements. Each element requires different methods of procedure in its extraction and subsequent recovery, and often it will cost more to recover an element than it is worth. For example, zinc is very troublesome, especially when chemically combined with iron. If the gold and silver also are with the iron, considerable difficulty will be experienced in getting any of these products out in a marketable form.

Another quality, which is a negative one, is a distinct lack of values in the ore to begin with. An ore is not necessarily worth treating because the total values foot up to ten or twenty dollars per ton. The aluminium from a clay carrying twenty per cent alumina would have a market value of fifty dollars, yet some difficulty would be encountered in deriving a profit from the treatment of such an ore. A true roscoelite carrying five per cent vanadium pentoxide might be said to have a value of $120 per ton, on the basis of a finished ferrovanadium, yet it would cost more than this to extract all of the vanadium and produce a marketable product from this class of ore.

5.—The problem is so indefinite, if indeed any problem exists, that it cannot be predicted which method of treatment will finally solve it. A shrewd guesser might do well to say electrometallurgical, because when taken in its broadest sense this term includes the other two.

6.—A microscopical examination would reveal the extent of mineralization, the size of crystals, and the general relation of the component parts of the ore. It is of great value to know what elements of the ore carry the values, and what its general physical conditions are.

I am of the opinion that complex ores which occur in sufficient quantities have a reasonably high metallic content, and show a fair paper profit after deducting standard costs and charges with a recovery of 50 or 60 per cent, will not have to wait long for a suitable method of treatment.

Canonsburg, Pa.

S. A. Ionides.—1.—The best answer I can give to the question "Is there a complex-ore problem?" is to quote from a letter recently received regarding an ore assaying about 5 oz. silver per ton, 14.5 per cent lead, 13 per cent zinc and 0.25 per cent copper.

"I note our recent shipments of ore are running exceedingly high in zinc, frequently reaching 17 per cent. The lead content remains normal, but we are having the life penalized out of us because of the increase in zinc."

As long as a valuable metal is a source of penalty, instead of being an asset, there can be no doubt about the existence of a complex-ore problem.

Another phase of the problem is the loss in slime-tailing. Two such problems have recently reached me. In one case the slime tailing assayed 4 per cent lead, 2 per cent zinc and 2 oz. silver per ton. It constituted 20 per cent of the original feed to a concentrating mill which is regarded as doing excellent work.

Another slime tailing from a mill where the work is not so high class contained 17 per cent lead and 11 per cent zinc. Undoubtedly a lower tailing could be obtained from the latter mill by straight concentration, followed perhaps by flotation, but even so, the product would have a low economic value.

A complex ore may be defined as ore containing two or more base metals. In its simplest form such an ore need not give trouble. For example, a copper-lead ore

containing precious metals is easily treated. A more limited meaning frequently is implied by the term "complex."

It involves the presence of zinc in the ore; and no instance is known to me where high recoveries of all metals are made from an ore containing lead and zinc, copper and zinc, or all three.

2.—Such ores as the one just mentioned have not proved amenable to treatment by present processes, in the sense that extractions exceeding 90 per cent of all metals can be made. It might almost be safe to say that this standard is never reached, for even if concentration was efficient, the smelting losses would reduce the total recovery below that figure. That there is a need for new treatment processes is evidenced by the fact that some of the large companies handling comparatively simple ore, are now engaged in elaborate experiments on new methods, particular reference being made to the copper leaching plant at Anaconda.

3.—From the number of inquiries received for methods of treating complex ores, there can be little doubt that a number of large mines require improved metallurgical processes. If the low-grade (10 per cent to 20 per cent) zinc carbonate ores of Leadville are admitted into the complex class, an estimate of "millions of tons" would be safe.

4.—The element that contributes most to the refractoriness of ores is zinc, and the quality or characteristic is friability.

A good commercial product can be made from ores containing copper or lead with precious metals; but inasmuch as a recovery of over 80 per cent is rarely made in practice, it may almost be said that these ores are refractory. The loss, in most cases, is due to slime; and the slime product from a lead or copper ore certainly is refractory.

The problem of concentration of coarse grades has practically been solved, and there is not much opportunity left for increasing the saving or reducing the cost.

Even so, there is room for improvement; for the miner still has to stand smelting and refining as well as concentrating losses.

The presence of zinc in an ore gives trouble all the way down the line, and if recovered with the lead it becomes a liability and not an asset. This point is accentuated as attention is now being directed largely to the solution of this particular problem.

5.—A solution of the problem along purely mechanical lines seems doubtful. Slime that will pass 150-mesh screen is invariably made in grinding, and the amount thus made frequently amounts to 20 per cent of the mill feed. The metal contents of this slime—gold, silver, copper, lead or zinc—usually will equal, if not exceed, that of the original ore. The only way this slime is being handled at present is by flotation, and while this process does efficient work where only one mineral is involved, it will not separate two. At best, one may be recovered; the other lost.

This year much work is being done along hydrometallurgical lines, but at the time of writing no method has been demonstrated a complete success. The most hopeful methods involve leaching with sulphuric acid, for copper ores, and a chlorination treatment for lead and zinc.

The electric furnace has proved its value in refining steel, making alloys, aluminium, etc., and there would seem to be a good field for it in ore treatment. This may be the method of the future, but at present it has not advanced as far as hydrometallurgical methods, combined, if necessary, with electrolysis as a final step.

Denver, Colo.

Program of Troy Meeting, American Institute of Chemical Engineers

The sixth semi-annual meeting of the American Institute of Chemical Engineers will be held at Troy, N. Y., from Wednesday to Saturday, June 17 to 20, 1914. The technical sessions will be held at the Pittsburgh Building of the Rensselaer Polytechnic Institute and at the Hotel Rensselaer, the latter being the hotel headquarters.

There will be technical sessions on the morning and evening of Wednesday and on the morning and evening of Friday. The following papers will be presented on Wednesday morning:

The Application of Physical Chemistry to Industrial Processes. By W. F. Rittman.

Studies on Filtration. By J. W. Bain and A. E. Wigle.

Scrubber for Vacuum Apparatus for Laboratories. By Chas. Baskerville.

On Wednesday night, Professor M. C. Whitaker will present his presidential address. This will be followed by an illustrated lecture by Dr. Wm. P. Mason on the Saratoga Septic Tanks.

On Friday morning the following papers will be presented:

Shoddy and Carbonized Waste. By L. J. Matos.

The Present Patent Situation. By M. Toch.

Bleaching Cotton Fibre. By J. C. Hebden.

Ozone in Ventilation. By J. C. Olsen and Wm. H. Ulrich.

The program for the Friday night session comprises the following two papers:

Development of Rotary Furnaces. By R. K. Meade.

A Combination Water Softener and Storage Tank. By L. M. Booth.

A great many visits and excursions are scheduled for Wednesday afternoon, the whole of Thursday, Friday afternoon, and the whole of Saturday.

On Wednesday afternoon the laboratories of the Rensselaer Polytechnic Institute and the U. S. Arsenal, where heavy guns are manufactured, will be inspected.

On Thursday morning the General Electric Co. plant at Schenectady will be visited and in the afternoon the filter plant at Albany, besides an automobile trip to Saratoga.

On Friday afternoon an excursion will be made to the plant of the West Virginia Pulp and Paper Company at Mechanicsville. On Saturday morning the Burden Iron Works will be visited where the puddling process for wrought iron will be shown. For the afternoon visits to the Cohoes filter plant (mechanical filters) and to the plant of the Freihoffer Baking Company are scheduled.

Among the social features of the Convention will be a subscription dinner at the Troy Club on Thursday night, a smoker at the residence of Dr. Mason on Friday night, and complimentary luncheons tendered by the Rensselaer Polytechnic Institute on Wednesday noon and by the General Electric Company on Thursday noon.

Professor M. C. Whitaker is President and Dr. John C. Olsen, Polytechnic Institute, Brooklyn, N. Y., is the Secretary of the Institute.

Two chloridizing plants are expected to be in operation this month. The first is that of the Northwestern Metals Co., at Helena, Mont., where experimental work has been going on for several years. The other is the Knight-Christensen mill at Silver City, Utah. The details of the process used differ, but the underlying principle is the use of chlorine to render soluble the base and precious metals.

Light from Unconsumed Electrodes

BY A. VOSMAER, PH.D.

The nature of the discharge which takes place between the electrodes of the arrangement that I shall describe presently has been puzzling every one who has watched it when demonstrated in my experimental laboratory in Holland. The discharge, causing an intense light, is absolutely steady, does not resemble any other, and does not cause any permanent change in the electrodes. In other words, one can obtain a light without consuming the electrodes, which seems something promising in the art of lighting. The discharge itself is not visible, which may be on account of the intense light at the electrodes. Occasionally one is able to distinguish a pale bluish-violet streak—straight, thin, tiny and faint—between the electrodes, but that does not seem to be the essential discharge which produces the effect of light.

The literature on discharges *in pleno* is not very elaborate. Lehmann, Toepler and Warburg have dealt with them at some length, and Lehmann especially has given full information about the subject, but the discharge I shall consider presently does not fall in the class of any

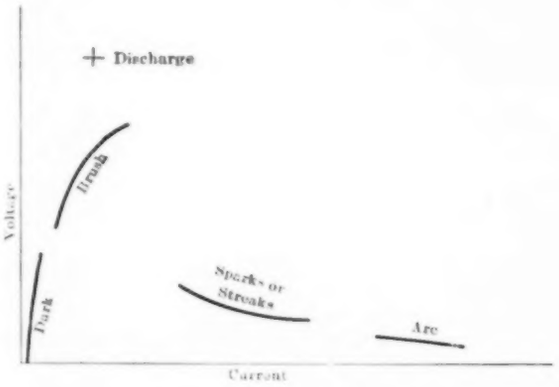


FIG. 1—DIAGRAM OF DISCHARGE

of the many discharges mentioned by these investigators. Let us first review the five fundamental discharges *in pleno*, and consider the diagrams given by Toepler[1] in discussing the question of boundary potential differences for different kinds of discharges. Toepler, using a sixty-plate influence machine, distinguishes sharply between the discharge from the positive and that from the negative electrode, and the graphs, Figs. 1 and 2, clearly show the difference in effect. We see a discontinuous line, indicating boundaries between the different discharges; first the non-visible, then the glow, the brush, the spark and the arc.

In alternating high-tension discharges, the current being supplied from a transformer, we must place the flame discharge between the spark and the arc. A static machine cannot give flame or arc discharges as transformers do, the current supply being inadequate and the working proper of the static machine ceasing when its electrodes are short-circuited, so to speak, by an arc discharge. The static machine is essentially a high-tension machine; but the flame and arc are essentially low-tension discharges. Consequently this machine is not adapted for the production of such discharges, but for the study of glow, brush and spark discharges it is better adapted than any other.

Toepler observed that in the actual production of the discharges there are no such things as definite boundaries, and he joined the lines by shaded parts. Fortunately he did not give different names to every part. In 1908 Cramp and Hoyle took up the matter again and confirmed Toepler's results. These investigators gave very interesting graphs for the relation of voltage to amperage, of which Figs. 3 and 4 are reproductions. They refer to the low-tension side and to the high-tension side of the transformer, and are an amplification of Toepler's lines.

If we consider the glow to be the beginning of *over-*

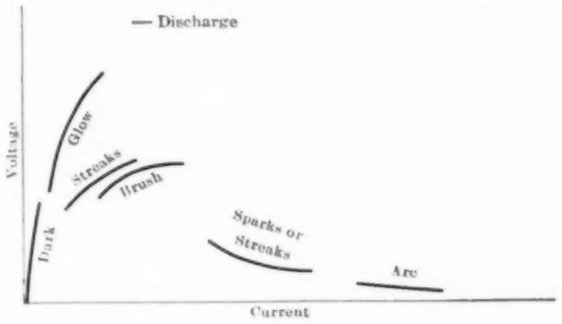


FIG. 2—DIAGRAM OF DISCHARGE

charge, and hence *discharge*, and the brush the period of regular gentle flow of electricity (from points), and the spark as the result of accumulated charge (from spheres), then the flame and the arc distinguish themselves from these in that they are low-tension phenomena, the others being high-tension discharges. If we take the flame discharge as being due principally to the conductivity of the gas at high temperature, then the arc may be considered as the controlled discharge due to transport of solid matter between electrodes. None of these ways of considering the discharges is correct, but they are given thus for the sake of reference.

Now the particular discharge under consideration could be called a high-tension arc discharge, because the potential difference between the electrodes is about 500 volts, as against about 35 to 50 for the ordinary arc; but since the arc light is accompanied by transport of solid matter, there is a difference between it and the light under consideration, in which nothing of the kind occurs. Neither is it a flame, a spark, nor a brush. The ill-defined discharge sometimes called "streak" is the only one most nearly resembling it. The peculiar features of the discharge in question are its high tension, viz., about 500 volts; its steadiness; the absence of electrode matter in the flame; the extreme temperature; its sensitiveness to polarity, as well as to frequency when alternating current is being used.

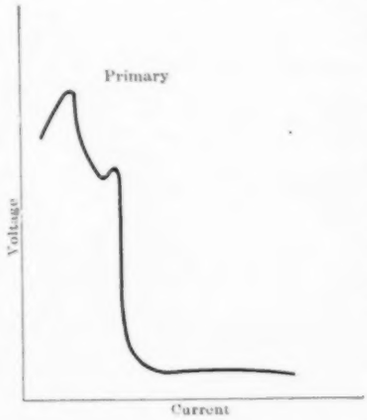


FIG. 3—VOLT-AMPERE CURVE

What is it? That is the question asked many a time, but as yet unanswered. Therefore, it may interest the reader to hear more about it. The production of this very peculiar discharge is simple indeed for those who are accustomed to making experiments, and who have that inborn sense of adjustment of conditions that distinguishes the true experimenter from the haphazard investigator.

[1] Ann. d. Phys. 1902, 7, 477.

As the question of polarity plays an interesting rôle in the phenomenon, it is best to take a good-sized Rhumkorff induction coil, preferably equipped with a modern Wagner hammer, allowing of close regulation. We might remind the reader that the induction coil fed by an interrupted primary current will give an alternating secondary electromotive force, due to the difference between the make and the break; but it should also be remembered that the value of said secondary tension depends, for one thing on the *rate* of appearance and disappearance of the lines of force. Now the great self-induction of the coil prevents a rapid "make," but the "break" can be very rapid, indeed, if proper means are provided. A condenser of right size is very important.

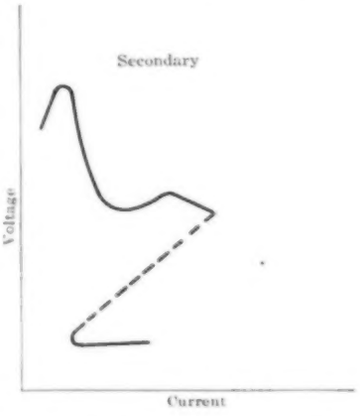


FIG. 4—VOLT-AMPERE CURVE

The great difference in time required for the make and the break, taken electrically, of course, causes a very marked difference in the magnitude of the secondary tension, so marked indeed that we may consider the secondary output to be practically an unidirectional, though intermittent, current.[2]

In Fig. 5 we give a representation of these considerations; the actual rotary mirror photographs look a bit different, but this diagram is true in principle.

As we shall refer to this point again, it may be well to say here that as the difference in polarity of the secondary terminals of a Rhumkoff coil is caused by the great difference in the time of appearance and of disappearance for the magnetic lines of force, the difference can be changed by variation in interruption, as regards speed of working, etc.

Suppose we have this induction coil as we want it, then all we have to do is to join the secondary terminals to an adjustable spark-gap allowing of rapid adjustment of polar distance of electrodes. Now take two bits of pure nickel wire, fix these in said electrodes, and all that remains to be done is to so regulate the different factors that the effect is obtained. The polar distance between the ends of the two nickel wires should be about 2 millimeters to start with. On setting the vibrator to work the ends will burn and form globules. Now adjust (while working) the polar distance until it is about 3 or 4 mm., and when once formed the discharge will go on for any length of time without further adjustment of electrodes. This is one proof that they are not consumed, for if they were the polar distance would be increased and there would not be a permanent condition.

As it is, no such thing as burning occurs. The discharge, when once properly established remains absolutely steady and can be watched and admired. We used to observe it in the cross wires of a telescope and never noticed change of place.

A more stringent proof of non-combustion is that in the spectroscope nothing but a pure continuous spectrum is to be seen, such as belongs to incandescent solid matter. No trace of lines is observed. If there were any transport of nickel, the vapor would show in the spectrum. Now some experimenting has to be done in order to get the right size of wire, corresponding to the size of the coil. My coil was a 12-inch, or a 4-inch spark length; the nickel wires were about 1 mm. for the first and 0.6 mm. for the second.

The explanation of the phenomenon is not easy to give. Probably the surface tension of molten nickel oxide happens to be so great that it maintains a certain volume, and the bombardment of electrons causes it to be white hot.

If, instead of nickel wire we take platinum, the latter will melt rapidly. In fact I have been able to melt or sinter almost anything in this little crater. Such substances as lime or quartz melt easily, but some of the rare earths only sinter.

The mistake of considering a flow of electricity to go from plus to minus is so common that we no longer pay attention to it, everybody knowing it to be the reverse. The best illustration is given by an ordinary carbon arc, wherein the positive, hollowing out to a crater, cannot very well be regarded as emitting particles. It was Armstrong who, in his beautifully illustrated work, "Electric Movement in Air and in Water," showed that current conduction is accompanied by mechanical movement of the conductor, if possible, and in the direction of negative to positive. So we are used to considering the negative electrode as the active, or discharging one, and the positive as the receiving department.

I have more than once drawn attention to the fact that in case we are dealing with true brush discharges it is *not* the negative but the positive electrode that discharges. Warburg holds the opposite view, but Toepler, Przibram, Cramp and Hoyle do not. In the present case, again, it is not the positive electrode that emits the light, but the negative; the positive being quite dark.

This anomaly may be emphasized. The case we first consider is that the electrodes are of the same material and the same size. If the sizes are unlike, say the negative a thin wire and the positive a thicker one, then we do not reverse the light emission but simply have a poor effect.

The effect, as stated, is shown best of all by pure nickel; next come some of the nickel alloys; but other metals do not exhibit the phenomenon in the proper way.

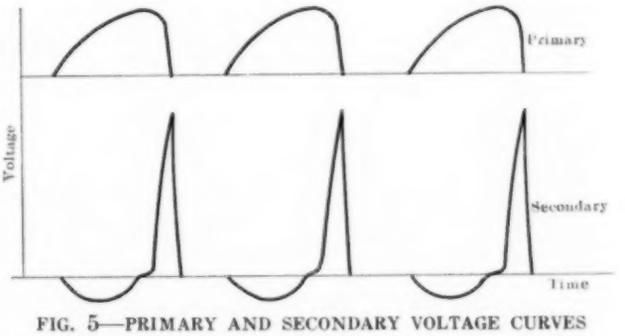


FIG. 5—PRIMARY AND SECONDARY VOLTAGE CURVES

When we feed the induction coil with alternating current or take an ordinary transformer, the effect is that instead of only one of the nickel electrodes glowing, both nickel electrodes will be glowing, as might be expected. But when we now substitute a copper wire for one of the nickel wires the result is that one, viz., the nickel electrode, glows, while the other, the copper electrode, is absolutely dark.

Remembering now what was said about the character of the induction coil's secondary current, we can make another interesting experiment. Upon slowing down

[2] Polarity of terminal is easily ascertained by having a strip of moist litmus paper on a strip of glass and holding this between the terminals; the plus shows in red as sharply as the minus in blue.

the speed of the vibrator (and it is just for this purpose that a hammer interrupter was preferred over any of the more modern kinds) the difference in the rate of appearance and disappearance of lines of force will also fade away, and there comes a moment when the unidirectional character of the secondary tension is lost altogether. At that moment, or period of transition, both of the nickel ends will light up, and on further slowing down it will be possible to have only one of them glow again, but not the same one as in the beginning.

We must leave it undiscussed whether or not this is due to a change in preference for the negative electrode to glow or to some other reason. The question is very complicated and instead of adding to the elucidation of the difficult problems in discharges it only adds more mystery. As to the commercial side of this phenomenon, it must be remarked that at present the experiments have been made on a very very small scale. Primarily it is a matter of scientific interest; but if we consider the possible value of a light from unconsumed electrodes, whereby all the trouble and expense of trimming is done away with, it seems worth while to investigate the matter on a larger scale and I hope to be able to do so when opportunity offers. For the present, the object of publication is only to communicate a remarkable and unusual discharge phenomenon.

341 St. John's Place, Brooklyn, N. Y.

Factors Affecting the Cost of Copper

In the annual report of Phelps, Dodge & Company for 1913, Dr. James Douglas, president, makes the following pertinent remarks relative to the company's operations in Arizona, New Mexico and Mexico:

"Judging from the average stability of price during a period of years we may infer that the price of copper in the future will not be lower than realized during the expiring decade. In fact, the price of the future should be higher to compensate for the increase in expense of producing the metal.

"Under legislative enactment and social aspirations the demands of labor will be more exacting, and the taxes, representing the cost of government (municipal, state and federal), will become more onerous. The mines, being always under corporate management, will have to bear more than their share of the burden. The state taxes paid by this company and its subsidiaries have risen from $431,559.76 in 1912, to $605,-431.78 in 1913, and federal taxes, amounting to $178,-872.84 based on 1913 business, have been levied. The cost of concentrating and smelting can probably be lowered by improvements in both methods and practice, but, with greater depth and the general decline in the grade of ore, the cost of mining will tend upwards rather than downwards; and the inevitable exhaustion of the company's mines must be anticipated and provided for by acquiring additional mining property. With this in view the mines of the Burro Mountain and the Chemung Company in New Mexico have been bought, and additions have been made to the property of the Detroit Copper Company, whose visible reserves have distinctly fallen off. This, fortunately, has not been the case in the Copper Queen and the Moctezuma Copper Company's mines, where the increased activity has resulted in largely increased tonnage of ore in sight."

The **P. Wall Mfg. Supply Company**, of 718 Preble Avenue, N. S., Pittsburgh, Pa., has sent us their new, well illustrated catalog on "Wall's Everlasting Steel Specialties," especially brazed steel oilers and torches, steel gong bells, galvanized steel ice cans, sheet metal specialties, copper-plated oilers and torches, etc.

Tropenas Converters vs. Electric Furnaces for the Manufacture of Steel Castings

BY G. MUNTZ

No one will begrudge the Tropenas converter the honor of having been among the early pioneers who made the steel casting business one of the most important in the world. Steel men are too broad minded to forget those who had the courage of their convictions and made it possible to obtain to-day a product which has become a decided factor in the manufacturing community.

It seems, however, that the fact has been overlooked that although steel as found in channels, I-beams, rails, etc., has almost the same characteristics as the steel that goes to make steel castings, the peculiar conditions inherent to the two manufactures are absolutely different.

We all know what the Tropenas converter can do and it is only necessary to state the claims of the electric furnace used to-day to manufacture steel castings:

1. The metal obtained is lower in phosphorus and sulphur contents than converter steel.
2. Segregation in electric steel is less than in converter steel.
3. Starting with cold steel scrap electric steel is cheaper to produce than converter steel.
4. Absolute uniformity.

We shall not take these claims and criticise them one after the other. We will simply draw a parallel between the Tropenas and the electric process and point out the obvious conclusions as we go.

RELIABILITY OF THE TROPENAS PROCESS

It is well to define here what we call the Tropenas process. Side blowing is an old idea, and even Bessemer, in his converter at St. Pancras, was blowing through the metal on the side and near the bottom. Adamson, Clapp, Griffith, Witherow, Walrand, Robert, all blew on the side of their converters, but each farther from the bottom. We believe that Tropenas was the first to blow on the surface of the metal or better between the slag and the metal. This principle we think characterises the Tropenas process.

This point being now clear, we want to emphasize the extreme reliability of the Tropenas converter, even when operated by a seemingly inexperienced steel man.

We all know that the Bessemer process and its derivatives alike are first of all ideal means of decarburizing pig iron. A converter cannot be better compared than to an oil lamp. As long as the oil is plentiful, the lamp is burning brightly; when the oil gets low, the flame begins to flicker, and when the oil is almost gone the flame drops. In the converter we burn mainly carbon, a flame ensues which will flicker when the fuel is almost gone and will drop when carbon exists only as traces in the final bath. With a converter the signs indicating the end of the operation are absolutely unmistakable and the blowing itself is so simple that operators can be trained to be most reliable in a very short time. Steel making, by the converter process, depends mainly on good cupola practice whereby suitable metal is secured for refining in the converter. This feature is a decided advantage and must be taken into careful consideration when comparing the Tropenas process to any other.

The electric process, like the open hearth and other similar methods, entails the use of closed furnaces, in which no outward signs are in evidence when the steel has reached the suitable composition. Test bars must be secured at various intervals and the quality of the steel determined in various ways or by direct examination of the fracture.

It is easy to see that the personal equation plays here an important part, and that very experienced men must be secured to properly melt and refine in an electric furnace.

THE ELECTRIC FURNACE AS A MELTING APPARATUS

Expert melters are scarce and command good salaries. Inasmuch as the electric furnace cannot compete with the basic open-hearth process, as far as cost of production is concerned, and is therefore limited to the small capacities, the resulting business ventures are comparatively small and do not admit of paying the large salaries which the big steel companies can allow to their experts. This is why the electric furnace has adopted the method followed by the old crucible process, whereby the furnace is used as a melting apparatus pure and simple.

The electric furnace has over the converter one great advantage, it can use such material as boiler punchings, turnings, borings, etc., which could not be melted in a cupola, meaning that although it is not as efficient a melting furnace as the cupola it can use cheaper raw material and thereby offset the loss in efficiency to its advantage.

In entering this field, the electric furnace, however, seems to violate the ever-present law of supply and demand. The recent stagnation of the steel scrap market caused such a great difference in prices between steel scrap and pig iron, that temptation might overcome better judgment resulting in the use of more electric furnaces, and consequently increased demand of such steel scrap as cannot be used in the converter process. Statistics show that although steel scrap is to-day at the same level as it was ten years ago, it has fluctuated very widely and often reached such levels as would have rendered the electric furnace a prohibitive melting method.

When determining the cost of liquid metal we find that the factors to be considered are well defined in the converter process, while in the electric furnace the main item, namely, the electric power is a very uncertain quantity. We will disregard the actual amount of current needed to melt 1 ton of steel, as opinions and experiences differ on that score, but there must be considered the cost per unit of the current itself. Steel foundries have given up the idea of manufacturing their own power. The investment required is too great compared to the returns to be expected, and the service offered by the electric companies is so superior that in most cases it is adopted. This advantage is offset by the fact that current is sold according to a sliding scale whereby rates decrease as consumption increases. It is found that in most cases the operation of an electric furnace is prohibitive when the number of operations does not allow the purchase of current at the favorable rates of the scale.

In other words, while a converter can be operated only once without great increase in cost of production, the same cannot be done with the electric furnace except at great loss.

This fact is worthy of consideration especially when one recalls the curtailed outputs which have prevailed for quite some time in the steel casting business.

UNIFORMITY OF PRODUCT

It is a well-known fact that while melting in an electric furnace the phosphorus and sulphur contents will not increase, but so far as steel castings are concerned these two elements are not of such capital importance. This, however, cannot be said of the carbon contents of electric steel which are not controlled as easily as in the Tropenas converter. In the converter, with operations stopped always at the proper time, absolute uniformity is realized and steels of any carbon contents can be obtained at will by the use of suitable additions. Table I of fifty consecutive blows in two Tropenas baby converters shows what can be expected from the pneumatic process.

In the electric furnace, final additions of ferromanganese and ferro silicon are made to either obviate oxidation of the metal or to carburize the steel to such an extent as will render it better suitable for the purpose intended, or both. The final product will then contain more carbon than the raw material first used, and inasmuch as at least 35 per cent of the steel poured is returned to the furnace in the shape of gates and risers, the carbon contents will keep on increasing, if delicate refining is not resorted to. However, if the percentage of risers and gates remains about constant, the carbon contents will decrease but only by decreasing increments and will never go over a certain limit.

The obvious conclusion is that the electric furnace as a melter will entail variations in carbon, which will occasion very noticeable changes in the physical properties of the steel produced.

TABLE I—TESTS FROM TWO TROPENAS BABY CONVERTERS BLOWING ALTERNATELY

Cast No.	Si.	Phos.	S.	C.	Cast No.	Si.	Phos.	S.	C.
11447	.26	.050	.048	.220	11489	.23	.048	.040	.208
11448	.27	.055	.047	.225	11490	.23	.046	.049	.244
11449	.26	.045	.046	.208	11491	.23	.048	.048	.220
11450	.28	.048	.050	.215	11492	.25	.048	.049	.215
11451	.29	.050	.048	.220	11493	.23	.050	.047	.228
11452	.26	.056	.054	.214	11494	.24	.046	.047	.217
11453	.30	.048	.046	.218	11495	.31	.045	.050	.220
11454	.26	.050	.046	.218	11496	.32	.050	.054	.212
11455	.27	.056	.048	.224	11497	.28	.050	.050	.206
11456	.28	.060	.053	.240	11498	.32	.046	.048	.210
11457	.30	.054	.048	.228	11499	.33	.048	.049	.212
11458	.28	.050	.046	.214	11500	.31	.046	.047	.210
11459	.28	.060	.056	.200	11501	.31	.047	.047	.210
11460	.27	.050	.051	.228	11502	.27	.050	.054	.227
11461	.31	.056	.050	.215	11503	.27	.055	.049	.190
11462	.28	.055	.047	.200	11504	.27	.055	.049	.190
11463	.27	.050	.046	.204	11505	.23	.048	.045	.218
11464	.26	.098	.044	.202	11506	.26	.040	.045	.218
11465	.28	.050	.044	.210	11507	.25	.043	.049	.198
11466	.27	.055	.050	.224	11508	.25	.043	.049	.190
11467	.27	.058	.052	.216	11509	.26	.043	.045	.215
11468	.30	.050	.048	.220	11510	.27	.044	.044	.230
11469	.31	.050	.044	.215	11511	.30	.044	.043	.238
11470	.30	.056	.049	.214	11512	.28	.045	.046	.206
11471	.31	.048	.045	.198	11513	.28	.045	.046	.206
11472	.30	.050	.045	.217	11514	.32	.045	.045	.205
11473	.32	.050	.047	.200	11515	.30	.045	.050	.238
11474	.25	.054	.045	.228	11516	.27	.044	.046	.238
11475	.25	.054	.048	.198	11517	.25	.042	.045	.217
11476	.20	.051	.045	.210	11518	.26	.042	.045	.230
11477	.27	.053	.049	.196	11519	.23	.042	.045	.232
11478	.29	.050	.044	.214	11520	.30	.042	.053	.237
11479	.30	.060	.053	.202	11521	.31	.046	.049	.212
11480	.31	.058	.050	.196	11522	.30	.046	.049	.212
11481	.25	.064	.051	.190	11523	.31	.046	.049	.225
11482	.28	.055	.052	.195	11524	.29	.045	.050	.208
11483	.27	.051	.049	.194	11525	.28	.045	.050	.223
11484	.30	.050	.045	.226	11526	.27	.050	.055	.206
11485	.33	.050	.048	.216	11527	.26	.054	.046	.220
11486	.30	.048	.044	.200	11528	.26	.053	.053	.238
11487	.27	.046	.042	.224	11529	.28	.052	.054	.202
11488	.26	.048	.042	.192	11530	.26	.056	.049	.226

Average manganese .80 to .90%. Variations in analysis are due to different weights of metal blown without altering weights of final additions. Average weight of blow, about one ton; average time of blow, 12 minutes.

Tests made by Michigan Steel Casting Company of Detroit, Michigan.

CAPACITIES OF FURNACES

Most of the steel foundries to-day are jobbing foundries and are called upon to produce a great variety of castings, some of which are bulky but made of thin sections, others bulky and heavy, others again small and weighing only a few ounces. To meet the peculiar exigencies of the steel casting business a furnace must, therefore, be flexible and able to produce metal of such temperature and in such quantities as will be suitable for any class of castings. Proper temperature can be obtained in both the electric furnace and the Tropenas converter and either process can produce castings which could not be poured successfully otherwise. Very hot metal can be had at no additional cost in the converter at slight additional cost in the electric furnace. The whole problem reduces itself to one of furnace capacity.

The electric furnace has the advantage over the converter to allow control of the temperature of the metal. This feature is valuable when metal has to be kept in the furnace for some reason or other; once the correct temperature is reached, it can be maintained at slight additional expense.

Metal can be held in the converter but not indefinitely as in the electric furnace. This brings out the fact that the electric furnace is limited in capacity to its own charge. In other words, a 2-ton electric furnace will produce 2 tons of steel every two and one-half or three hours.

If we compare this to what can be done in a converter of equal charge capacity we find that from two to three operations can be made per hour, thereby allowing the accumulation of from 4 tons to 6 tons of steel per hour. We recall the instance of an 18,000-lb. casting being poured successfully with a single 2-ton converter. Six blows were made to secure sufficient metal, each blow being poured on top of the other in a ladle; it took one hour and forty-five minutes to make the steel and after the casting was poured there was not even a skull left in the ladle. This shows that even after waiting almost two hours the steel was still very hot.

This feature of rapidity of operation is one of the great factors in favor of the converter. It reduces itself to the fact that even should an electric furnace be as cheap of installation as a converter of equal charge capacity (which is not the case) it would take five or six 2-ton electric furnaces to produce the work of one 2-ton converter.

SEGREGATION

The various impurities which are found in steel have different freezing points, and show a tendency to segregate and concentrate in certain portions of large bodies of cooling metal.

This segregation is affected by various factors such as initial temperature of the molten mass and especially the rate of cooling. From experiments made by Talbot in 1905 it was concluded that sulphur segregates most, phosphorus next, then carbon, and finally manganese. It was found also that the usual additions of aluminium reduced the segregation considerably.

Authorities differ as to the elements which segregate first, and Professor Howe shows that carbon segregates the most readily.

It is therefore interesting to apply either results to the particular case on hand.

The field covered to-day by the Tropenas converter and the electric furnace is such that metal manufactured by either process freezes almost instantly after being poured. Small and medium steel castings offer numerous surfaces of cooling and even if the steel produced by the converter process contains more sulphur and phosphorus than electric steel, segregation will take place only in negligible quantities as the rate of cooling is too great to allow any extensive concentration.

On the other hand, if we adopt the conclusions of Professor Howe, segregation will be greater in electric steel inasmuch as carbon has an automatic tendency at being higher than in converter steel.

INFLUENCE OF LARGE SULPHUR CONTENTS

It is a well-known fact that with sulphur contents up to 0.1 per cent the direct action on ductility and tensile strength is negligible. The only detrimental effect of sulphur is to cause red shortness, or cracking and checking if the steel has to be forged or rolled.

Neither the converter nor the electric furnace contemplate, to any great extent, the manufacture of steel castings which will later be forged. It ensues that higher sulphur contents in converter steel would not prove detrimental to steel castings especially if the manganese contents are properly graduated to offset any possible bad effect due to sulphur.

The only advantage of very low sulphur therefore seems to apply to the rare case of steel castings which have to withstand high temperatures.

INFLUENCE OF LARGE PHOSPHORUS CONTENTS

Phosphorus is the element which is the most detrimental to steel as it creates brittleness under shock, and general cold shortness. It is the consensus of opinion that phosphorus contents should not exceed 0.08 per cent for all practical purposes. Between 0.05 per cent and 0.1 per cent the elongation and reduction of area are not affected to any great extent while the tensile strength increases. The readings of the testing machine are not therefore of any special value when the effects of phosphorus are considered. Cold bending tests, by use of a drop, are best suited to determine the quality of steel castings, but in the case of phosphorus we find results erratic and capricious and it seems that on this score the best factor to be considered after chemical analysis is the reliability of the producers of the steel.

In the present controversy it must be admitted that the electric furnace has the advantage of being able to use such raw material as will insure low phosphorus contents, but accumulation of phosphorus is very slow in the converter process and the chances of picking it up from fuel, while melting, are very remote. It is, therefore, just to say that converter steel while higher in phosphorus contents than electric steel, more than come within the ordinary limits of safety.

It has been regretted that there did not seem to be any ready market for electric steel castings. The arguments given in the present article might provide a good basis of explanation.

Tropenas Converter Co.,
New York City.

Award of the Willard Gibbs Medal to Ira Remsen

At a banquet of the Chicago Section of the American Chemical Society, on May 15, the Willard Gibbs gold medal was presented to Dr. Ira Remsen, a founder and former president of Johns Hopkins University. Founded by William A. Converse of the Dearborn Drug and Chemical Co., Chicago, the medal has been awarded before to three other prominent chemists.

In an address on "The Development of Chemical Research in America" Dr. Remsen spoke of the necessity of studying chemistry for the love of the science and commented at some length on results obtained by now prominent chemists who at one time had been his students. Among those at the speakers' table were President George E. Vincent, University of Minnesota, Professor Harry McCormick, Armour Institute of Technology, President Abram W. Harris and Dr. William A. Noyes, Northwestern University.

Electrolytic Zinc.—The Borchgrevink electrolytic process for the treatment of zinc ore is to be tried on a larger scale in Norway. Mr. Borchgrevink has succeeded in making his process a success with zinc ore in lime stone containing less than 6 per cent. zinc, so he has been working so far only on a small scale (200 kilograms per doz.). A factory is being built near Kristiania and is expected to be ready in September, with a capacity of one ton per day.

Paul Héroult

Paul Héroult, the famous pioneer of the electrometallurgy of aluminium and steel, died in Paris on May 9, at the age of only fifty-one years. He had been in poor health for the last few years, suffering from kidney trouble, yet to his many friends on this side of the Atlantic the news of his death came as a sudden unexpected shock.

Paul Louis Toussaint Héroult was born at Thury-Harcourt (Calvados, Normandy) on April 10, 1863. Since his grandfather lived in England he spent his youth's happy days partly in France and partly in England. He was educated at the Lyceum of Caen, Normandy, and later at St. Barbe in Paris, where he graduated in 1882, to enter the Paris School of Mines (Ecole des Mines). From 1883 to the beginning of 1884 he served in the French army and then returned to the School of Mines, but on account of the death of his father he was soon forced to quit his studies in order to conduct the business of his father's tanning works.

However, Héroult continued there his metallurgical researches. He was particularly interested in the electrochemical production of metals. He bought a small Gramme dynamo of 1.5 hp, which he replaced in 1886 by another small machine giving 800 amp at 15 volts. This machine was especially built for his aluminium process, for which he had applied for patents in the same year. In 1887 he became technical manager of the Neuhausen aluminium works of the Swiss Metallurgical Company, where his process is in use up to the present. He then started an aluminium plant in France, at Froges, Isère, for The Société Electrometallurgique Française. About the same time he made two trips to this country, where meanwhile, independently of, and simultaneously with him, Charles M. Hall had worked out what was essentially the same process for electrolytic reduction of aluminium and had started the works of the Pittsburgh Reduction Company.

THE LATE PAUL HÉROULT

After his return to France, Héroult assumed the management of the Froges works, but after the erection of a larger water-power plant in La Praz, Savoy, he became general manager of these new La Praz works. In 1892 the power developed at La Praz was 3000 hp, which was increased to 13,000 hp in 1895.

Héroult's aluminium furnace, which by the addition of a tap hole was rendered a continuous apparatus, became in his hands the instrument for numerous further researches. In it he made the first commercial calcium carbide produced in France. In it he undertook for the Société Electrometallurgique de Froges, the manufacture of artificial corundum, ferrosilicon and ferrochrome, and thus he was led to the researches on the use of the electric furnace in the iron and steel industry to which the balance of his life was chiefly devoted.

In 1904 Héroult came again to the United States to attend the St. Louis World's Fair, and since that time he resided alternately in this country and in France. The American rights of his electric steel furnace which made rapid progress in Europe, were acquired in 1910 by the United States Steel Corporation. With respect to the reduction of iron ore in the electric furnace, Héroult assisted Dr. Haanel's Canadian government investigation at Sault Ste. Marie and also helped in the early work of the Noble Electric Steel Company at the place named in his honor, Heroult-on-the-Pit in Shasta County, California. He later became consulting engineed for the new works of the Southern Aluminium Company, in course of erection at Whitney, N. C., but his failing health forced him to retire. He went to Paris, where he lived the last few years. He was married twice and leaves a widow and five children.

Especially through his connection with the aluminium industry Héroult accumulated a very considerable fortune. Many honors were bestowed on him; the Institute of Technology of Aachen, Germany, conferred on him the honorary degree of Doctor of Engineering; France gave him the red ribbon of the Legion of Honor and the Grand Lavoisier Medal. He had many ardent friends in this country and was a member of the American Electrochemical Society and The Chemists' Club. At the meeting of the New York Sections of the American Electrochemical Society, the American Chemical Society and the Society of Chemical Industry, held on May 22, Dr. C. A. Doremus spoke as follows in proposing a resolution in honor of Héroult's memory:

"Voyagers of old made new maps of the earth. Héroult, the pioneer in metallurgy and engineering, has staked out new fields for his successors to develop. The visions of linking the two great oceans across the Isthmus have only been realized after centuries, but in the span of a quarter of a century Héroult brought to successful practice two epoch-making discoveries. Nor did he rest content with these.

"He showed us how to release aluminium from the bondage in which it has been held for geologic eons; he utilized it in the arts; he gave it to us for the treatment of steel and then carrying forward his intelligent use of the electric current not for electrolysis this time, but for thermic results, has founded a new art in the purification and treatment of the metal for which his age is named.

"Héroult was original, imbued with the ardor characteristic of youth, he was intrepid, resourceful, persistent, and reduced his inventions to practice, in ways that brought him international recognition. And then what a man!

"Whoever had the good fortune to meet him must have been impressed with his modesty, yet also with his latent power; with his cosmopolitanism, yet pride in his native land; and above all with that genial nature that endeared him to his friends and to professional brothers."

A resolution was adopted conveying to the widow and children the most heartfelt sympathy of the members of the three societies and the assurance of their esteem and affection for Héroult as a man, their gratitude to him for the wonderful inspiration he gave to all, working as he did for science and industry.

An appreciation of Héroult as a pioneer of electrochemistry will be found on our first editorial page.

A Study of the Annealing Process for Malleable Castings

BY OLIVER W. STOREY.

This article gives the results of experimental work carried out by Mr. E. L. Leasman under the author's direction at the Chemical Engineering Laboratories of the University of Wisconsin. Although these experiments were preliminary in nature as a basis for extended work, interesting results were obtained.

Malleable cast iron is a product of importance. The annual production of the United States is over 1,000,000 tons. And yet, in such an important industry scientific progress has been slow. Each malleable iron plant makes its product under different conditions, and, as a result no two products are exactly alike except that all usually have to meet standard specifications.

The lack of scientific knowledge of this industry is made evident by the sparsity of the literature. The only book on this industry known to the writer is Richard Moldenke's recent work, "The Production of Malleable Castings." In the preface he states, "of the closely held secret processes of the great iron industry none can rank with the production of Malleable."

The first record of the process is by Reaumur, in 1722. It is probable that the malleablizing process was practiced much earlier, but, no doubt, it was like so many of the earlier industries, and was a secret which was handed down from father to son.

The pioneer manufacturer in this country was Seth Boyden, of Newark, N. J., who, although he did not discover the underlying principles involved, recorded his experiments. These are the first we have and he deserves special credit since they were made at a time when scientific experimentation on industrial problems was frowned upon.

Malleable iron occupies a position between ordinary cast iron and cast steel. It has a higher tensile strength than cast iron, but not as high as that of cast steel, while its resilience is superior to either.

The method of producing malleable castings is fairly uniform throughout the country. The white iron castings are packed in rolling mill scale in rectangular containers. These containers, either three or four high, are carefully luted up to exclude currents of air, placed in the ovens and heated to the desired temperature, being kept at the high point at least sixty hours. The total time of the annealing process is a week from one time of "lighting-up" to the next time.

The experimental work was carried out to determine the effect of the following variables upon the annealing process.

1. Packing materials.
2. Temperature of annealing.
3. Time of annealing.
4. Rate of cooling.

This was undertaken as a metallographic problem. All specimens were studied with the microscope which showed the structural changes.

In making malleable cast iron, white cast iron is used for the hard castings. This white iron must analyze to within certain limits to insure proper annealing. It is then annealed at the proper temperature, usually between 750° and 900° C., depending upon the practice, for about 60 hours, after which it cools slowly.

The structure of such a white cast iron is shown in Fig. 1. The two constituents shown are the white cementite and black pearlite, which is an eutectoid of ferrite and cementite. The cementite or iron carbide, Fe_3C, is present in two forms; first, as excess, and second, associated with ferrite as pearlite.

After being annealed at the proper temperature and slowly cooled the interior structure is usually that of Fig. 2, ferrite and temper carbon, with an outer shell that may vary from a pure iron to a high carbon steel. Fig. 3 is the fracture of a test lug, showing the black heart with an outer bright steel rim.

It is evident that the formation of the malleable structure of ferrite and temper carbon is due to the breakdown of the iron carbide into its elements, iron and carbon, at the annealing temperature, according to the following equation:

$$Fe_3C + \text{heat} = 3Fe + C$$

This reaction explains the formation of the black heart but not the rim structure. A different condition is necessary to produce a pearlite shell than to produce one of pure ferrite. Three different rims are shown in Figs. 4, 5 and 6. Fig. 4 is the rim structure of the fracture shown in Fig. 3, and shows a medium carbon steel area covered by a thin skin of ferrite. Fig. 5 shows an outer rim of nearly pure pearlite, while Fig. 6 shows one of pure ferrite.

The four variables mentioned determine the structure of the casting. The packing materials will affect the rim, while the time and temperature of annealing and rate of cooling will determine the interior structure, with only a minor effect upon the rim.

The white iron used for the experiments was kindly furnished by the Beaver Dam Malleable Iron Company. The material was in the shape of test bars ¾ x ¾ x 10 inches. Mr. W. G. Grimer, chemist for the above company, furnished the following analysis with the bars:

Total (combined) carbon...	2.60 to 2.70%
Silicon	0.65 " 0.70%
Sulphur	0.055 " 0.060%
Phosphorous	0.140%
Manganese	0.27%

Fig. 1 is a micrograph of the white iron used. The structure shows a high percentage of pearlite owing to the low percentage of carbon resulting from steel additions to the melt.

The annealing furnaces used for these experiments consisted of two muffles electrically heated by resistance wire. A low voltage regulating transformer gave absolute temperature control.

All thermocouples used were calibrated frequently against a standard platinum, platinum-rhodium couple.

In order to carry on a larger number of experiments in this preliminary work a number of small containers were made. Several could be used in the furnace at the same time. These containers were made of 1½-inch pipe, cut long enough to take specimens 2 inches in length. They were then luted with fire clay except where caps and nipples were used. The serious objection to these small containers was that the oxygen of the air usually had access to the specimen.

Effect of Packing Materials

The first variable to be studied was the packing material. It had been noticed that certain specimens of commercial malleable iron had an enveloping skin of steel (Fig. 5), others pure carbonless iron (Fig. 6), while others showed an intermediate structure (Fig. 4). It was thought that the packing material would determine the structure of the outer shell.

A large number of packing materials were used, including lime, sand, alundum, rolling mill scale, resistor carbon, manganese dioxide, cement, fine and coarse fire clay, iron oxide, alumina, fine graphite and carborundum.

The results obtained by the use of these various materials showed that their physical and chemical constitution influenced the skin structure of the annealed casting. The packing material had no apparent effect upon the interior in the American "black heart" practice.

The chemical composition of the packing material was a factor that partly determined the nature of the outer portion of the annealed casting. With the materials which were stable at the annealing temperature the structure was uniformly a layer of ferrite. With oxidizing materials a deep layer of ferrite was

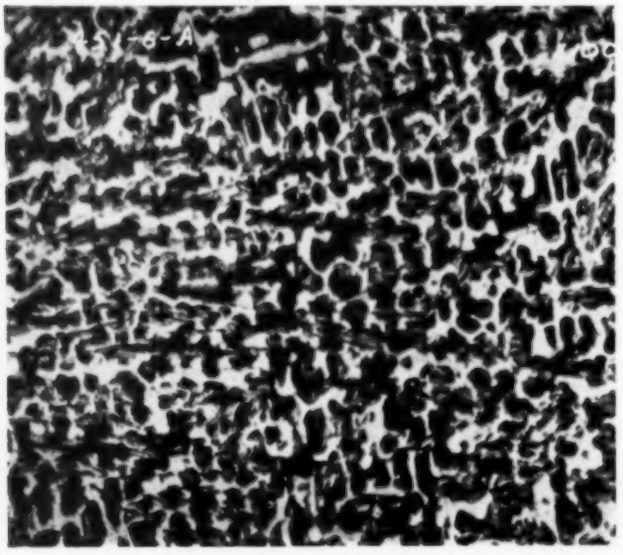


FIG. 1—MICROPHOTOGRAPH × 100

formed. It is probable that enough oxygen was liberated to decarbonize to a greater depth.

With carbon, charcoal, graphite and case carbonizing materials, a skin structure was formed which varied from a low-carbon to a high-carbon steel.

The physical structure of the packing materials also affected the malleable skin structure. Materials which were loose and porous and allowed a free circulation of the air showed a ferrite exterior. Without the oxygen of the air the structure of the exterior portion did not differ from the interior. This was best illustrated by the following experiments.

Brass turnings were used as a packing material in one of the containers. The brass fused at the annealing temperature and completely inclosed the specimen, preventing the oxygen from coming in contact with the exterior of it. Although a normal malleable structure was formed in the other specimens annealed in the same heat, the temper carbon extended to the brass coating which had formed on the specimen.

That the chemical composition of inert packing materials has little effect on the rim structure is evident from samples of commercial iron. Specimens from various plants using mill scale as a packing material showed a rim structure varying from pure ferrite to a high carbon steel. It is probable that the atmosphere of the furnace and the looseness of the packing rather than the chemical action of the scale determined the rim structure.

The experiments were not carried far enough to determine under what conditions the various degrees of carbonization could be obtained. Even if results were obtained these could hardly be comparable with those obtained in malleable practice. The atmosphere of the experimental furnace was highly oxidizing as compared with that of the commercial furnace. The small containers also allowed a quicker penetration of oxygen than those used in commercial practice. The probable exception was in the use of unusual materials such as case carbonizing mixtures.

But with materials such as mill scale, sand, clay, cinders, ashes, etc., the exterior structure will depend upon the composition of the atmosphere in the furnace, upon the annealing temperature, the method of packing, and the rate of cooling.

The writer recalls one malleable plant where scale is regularly used when the furnaces are running at the normal rate. But when a rush comes anything within reach is used as a packing material, usually ashes and cinders. No difference in structure is apparent between the two malleable products.

In another plant brick dust is replacing scale with no apparent change in structure. Large castings are annealed by merely heating in a muffle, care being taken to prevent excessive oxidation. The structure is the same in all cases, consisting of an exterior layer of ferrite, or pearlite, or both, and an interior of temper carbon and ferrite.

The skin structure, when a neutral atmosphere is used, does not differ from the interior since the breakdown is the same throughout the specimen. This was shown by the specimen annealed in brass. This shows that a slightly oxidizing action is necessary to secure a steel rim and that either a too highly oxidizing or a neutral atmosphere will give a ferrite or a ferrite and temper carbon rim, respectively.

The rate of cooling also has a slight effect upon the skin structure. A slow rate of cooling tends to result in a ferrite rim. This will be discussed later.

Where a case carbonizing mixture was used as the packing material the rim resembled the original iron (Fig. 11), since some of the excess cementite remained undecomposed. This structure will also be discussed later.

A peculiar rim is shown in Fig. 12. Here two steely strata are seen with a ferrite structure between them. Only one specimen showed such a structure.

The following conclusions may be drawn:

1. The packing material does not influence the interior of "black heart" malleable cast iron.

2. The ordinary packing materials do not directly affect the skin structure of malleable cast iron.

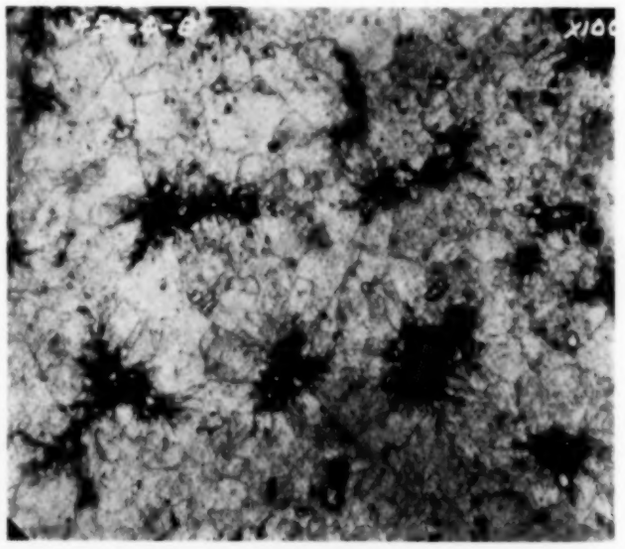


FIG. 2—MICROPHOTOGRAPH × 100

3. A loose packing material allowing a free circulation of oxidizing gases will result in a carbonless rim.

4. A slightly oxidizing atmosphere will result in a steely rim.

5. A neutral atmosphere will result in a rim having the same structure as the interior of the iron.

6. A slow rate of cooling will tend to result in a rim of ferrite.

Effect of Temperature of Annealing

A number of experiments were run to determine the most effective annealing temperature for iron of the analysis given, where the time of annealing was 50 hours and the cooling to 250° C. took 50 hours.

Experiment No.	*Annealing Temperature Deg. C.*
4	815°
5	760°
6	700°
7	730°
9	840°
10	790°–800°

Results

The specimens annealed at 700° were too hard to saw though slightly malleableized.

The specimens annealed at 730° were sawed with

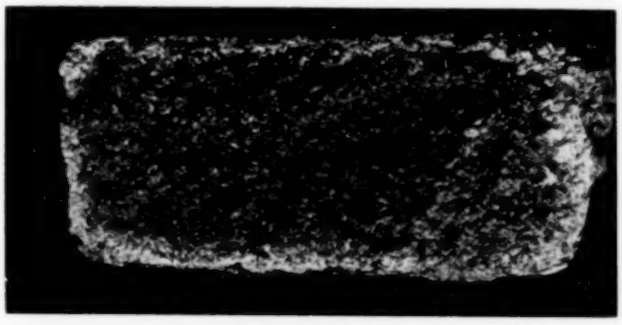


FIG. 3—MICROPHOTOGRAPH × 2½

difficulty. Upon examination with the microscope the structure had broken down slightly.

The specimens annealed at 760° showed a partial breakdown of the cementite, as shown in Fig. 7.

Specimens annealed in experiments 4 and 10 showed but a partial decomposition of the cementite.

Upon annealing at 840° the carbide was almost entirely broken down (Fig. 8).

In one experiment the temperature was kept at 535° to 650° F. for 120 hours, but these specimens were too hard to saw.

In the first experiments an annealing temperature of 870° to 900° was maintained for 50 hours, and this resulted in a complete breakdown of the cementite. (Fig. 6.)

The effect of the temperature is an important factor. Not only does the time of the anneal, but also the quality of the resulting product depend upon the temperature. It is possible to cut down the time of the anneal by raising the temperature, but woe befalls those who do it. A poor product results. The lower the temperature of the anneal the better the product. The balance must be found between the increased value of the improved product and the high cost of the longer anneal.

Figs. 2, 6, 7 and 8 are micrographs of structures of malleable iron annealed at varying temperatures. Fig. 7 shows the result of annealing at 760°. The micrograph shows areas of cementite in a background of ferrite with a few particles of temper carbon. The dendritic structure of the cementite is visible. This shows that the cementite in the pearlite suffers the first breakdown and that the excess cementite is the last to dissociate.

By annealing at 840° the structure of Fig. 8 was obtained. Small particles of undecomposed carbide are still visible though the specimen is composed mainly of ferrite and temper carbon.

Fig. 2 is a micrograph of some commercial iron and shows small areas of cementite. Most commercial malleable iron shows such areas, indicating that the annealing was not entirely completed.

In Fig. 6 the structure is entirely ferrite and temper carbon, indicating that the breakdown at 870° was complete.

Conclusions

1. With an annealing time of 50 hours the white iron furnished required an annealing temperature of 860° to secure a complete breakdown of the cementite.
2. Annealing at 740–760° for 50 hours resulted in a partial breakdown. By continuing the heat to double the above time it is probable that a complete breakdown could have been secured.
3. No decomposition could be observed in specimens annealed below 670°.

The Effect of Rates of Cooling from the Annealing Temperature to 250° C.

In all malleable iron work one of the most important factors is the rate of cooling. In the anneal the cooling is slow, usually requiring from 40 to 60 hours. If the cooling is too fast a hard, brittle, steely structure results.

In this research the result of the different rates of cooling was noticed in the first three experiments. When the rate was slow, that is, the drop from 900° to 670° required 38 hours, the interior of the specimen showed an excellent malleable structure, ferrite and temper carbon. (Fig. 6). When the same drop in temperature required but 22 hours, the resulting interior structure showed ferrite, temper carbon and areas of pearlite. (Fig. 9.) When the time occupied but 15 hours the interior showed pearlite, temper carbon, and a small amount of ferrite. (Fig. 10).

From these data it was evident that the structure of the resulting anneal was dependent upon the rate of cooling. It was not only desirable to determine the most suitable rate of cooling, but also to obtain the critical range through which this cooling had to be slowest.

The A_1 temperature, 700°, was regarded as the

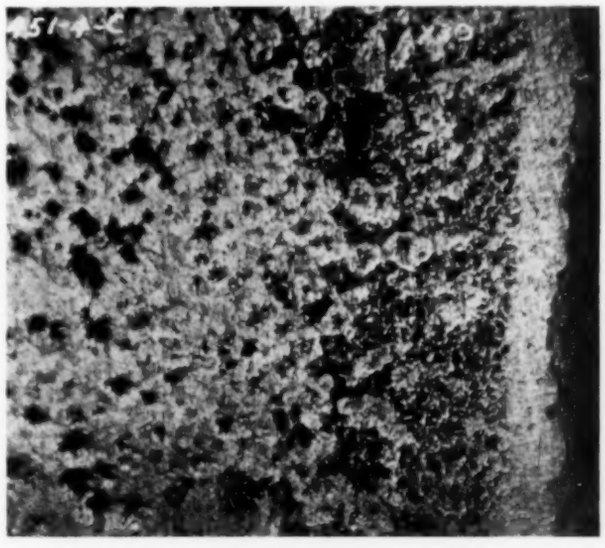


FIG. 4—MICROPHOTOGRAPH × 30

critical point. At this point the transformation of austenite into pearlite occurs. The experiments were run so that the critical range was above 670°, allowing for a lag of 30°.

In experiment No. 11, the first of this series, the furnace burned out and the time of cooling was rapid, requiring 14 hours to drop from 870° to 250°. All

specimens showed an interior of high-carbon steel and temper carbon similar to that of Fig. 10.

In experiments No. 12 to No. 16 the time of cooling from 870° to 750°, 750° to 670°, and 670° to 250° was varied.

From these experiments one conclusion was drawn. The rate of cooling from 670° to 250° had little, if any, influence on the structure. Though the evidence was

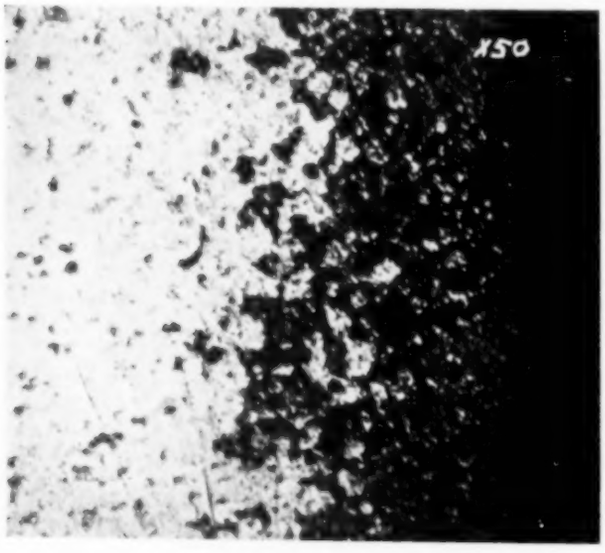


FIG. 5—MICROPHOTOGRAPH × 50

inconclusive the work done so far indicated that the rate of cooling from 840° to 730° determined the re-austenite will be precipitated in that range.

When the malleable iron made in these experiments was reheated for 10 minutes the carbon did not go into solution below 800° and form a steel with air cooling. A slight amount of carbon went into solution when heated for 1 hour at 800°. If the malleable iron was heated at 850° enough carbon went into solution in a few minutes to form a high-carbon steel. This experiment showed that the carbon could not precipitate out entirely above 800°, and that the critical range was below that temperature.

Specimens which had been heated to 900° and consisted of pearlite and temper carbon were annealed at 700° for various periods of time and cooled in the air. When heated to 700° for one-half hour part of the pearlite was decomposed and a narrow band of ferrite surrounded each particle of temper carbon. When heated for 2 hours about one-half of the pearlite was decomposed and the temperature resembled Fig. 10, though the outlines were not as definite. By heating the specimen for 5 hours at this temperature all of the pearlite was decomposed and structure again consisted of ferrite and temper carbon. This experiment would seem to show that the final breakdown of the cementite may occur above the Ar_1 point. At the present time the effect of heating at temperatures above and below 700° has not been studied.

It was noticed that particles of undecomposed excess cementite remained unchanged throughout these heat treatments showing that it was stable in comparison with the dissolved cementite.

If the temperature necessary to entirely decompose the iron, iron-carbide solution was below 800°, and if the reaction was negligible below 670° it was evident that a good structure could be obtained by holding the specimens at the critical temperature upon cooling for the correct length of time and then cooling at a quick rate. It was also evident that a good structure could be obtained by annealing at this critical temperature and cooling rapidly. This was done, and although the time required was longer, owing to the lower temperature, a good structure resulted.

The temperature of the final precipitation of carbon is probably dependent principally upon the silicon and manganese content of the iron. The lower the silicon content the lower the temperature at which the iron carbide is stable.* The proportion of these elements will determine the critical range of the iron.

Conclusions

1. A fast rate of cooling after annealing results in a steely structure.
2. The critical cooling range is between 740° and 830°.
3. If the rate of cooling is sufficiently slow in the critical range all of the carbon in the undecomposed austenite will be precipitated in that range.
4. If the carbon is entirely precipitated in the critical range the rate of cooling below 700° has little effect upon the iron.
5. Very slow cooling will result in a ferrite rim.

Time of Annealing

A number of experiments were run to determine the correct time of annealing at various temperatures. The results showed that a malleable structure was obtained in 36 hours at 870°. By raising the temperature to 985° the time was cut to 18 hours, while at 740° the necessary time was more than 70 hours.

Though the short, high temperature anneal looks attractive it is not practicable owing to several reasons. It soon ruins the furnace, burns the saggers, and results in a poor grade of malleable iron. The lower the temperature of anneal the cheaper the upkeep of the furnace and the better the iron. Though the best product results from the 740° anneal, this low temperature is usually prohibitive where cost must be considered, owing to the length of time.

Reactions In the Annealing Process

A metallographic study of the specimens showed the successive reactions occurring in the process. The fol-

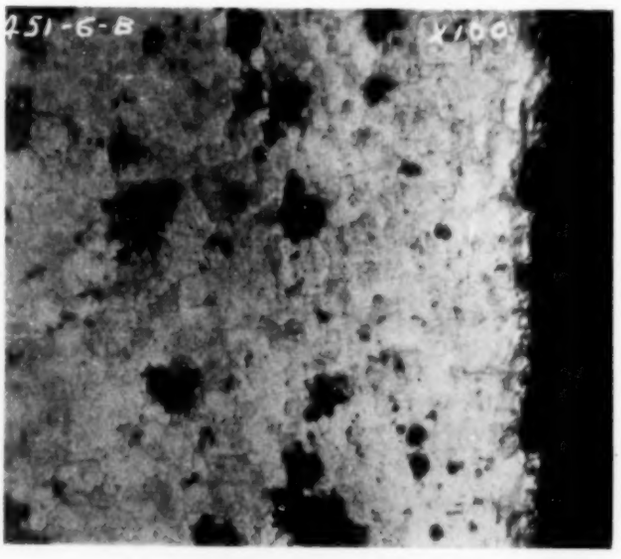


FIG. 6—MICROPHOTOGRAPH × 100

lowing data are significant in determining the reactions resulting in the malleable iron structure:

1. When the malleable iron was heated for too short a time or at too low a temperature and slowly cooled the structure consisted of ferrite, cementite and temper carbon. (Fig. 7.)

*Charpy and Cornu. Compt. rend. 157, 901.

2. By increasing the time at the maximum temperature or increasing the temperature the cementite areas were almost (Fig. 8) or entirely eliminated. (Fig. 6.) The structure then consisted of ferrite and temper carbon.

3. By annealing at the proper temperature to insure the breakdown of all of the excess cementite, but cooling rapidly from this temperature, the structure of Fig. 10 resulted. This consists of temper carbon surrounded by ferrite, in turn enveloped by pearlite.

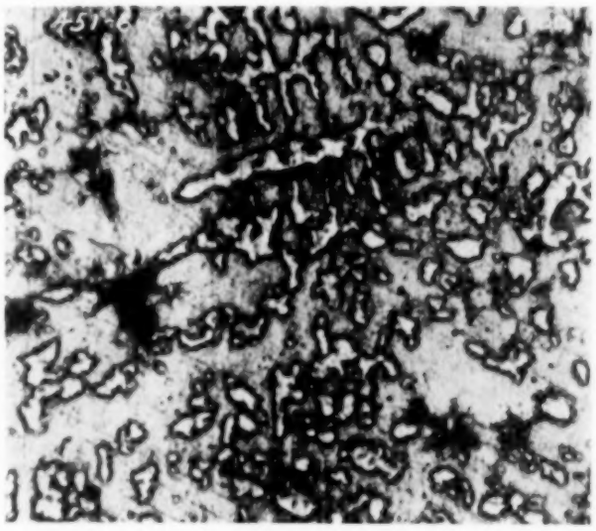


FIG. 7—MICROPHOTOGRAPH × 100

4. When the rate of cooling was slower than in No. 3 the structure of Fig. 9 resulted. This is similar to Fig. 10, but the pearlite areas are much smaller.

5. By annealing at too low a temperature and cooling rapidly the specimen consisted of areas of excess cementite surrounded by pearlite and small particles of temper carbon surrounded by ferrite.

6. When the malleable iron was reheated to about 800° the temper carbon started to go into solution.

7. It was possible to secure a complete decomposition of the dissolved cementite at 700°.

From these data the mechanism of the formation of the malleable structure could be deduced.

The white iron consisted of cementite and pearlite. When heated to the annealing temperature this pearlite went into solid solution as austenite.

It is a question whether the breakdown of the cementite occurs directly from the excess cementite or whether it occurs from the iron carbide dissolved as austenite. Though the research did not result in any definite conclusions, it seemed to indicate that the excess cementite did not break down directly, but that it first went into solution as austenite, from which the carbon was precipitated.

If the excess cementite of the white iron broke down directly the reaction should show some free graphite scattered in areas of cementite, since the decomposition would proceed uniformly throughout the carbide particles. Instead of a uniform breakdown and the scattered formation of temper carbon these cementite areas grow smaller as though they were being gradually dissolved, while the temper carbon forms in large patches. There is no indication of a direct decomposition of the excess cementite.

If the cementite in the austenite decomposes and deposits the temper carbon the ferrite liberated will dissolve the excess cementite. The presence of the nucleus of temper carbon probably causes a more rapid breakdown of the dissolved carbide.

In the specimens examined in which excess cementite remained owing to imperfect annealing, it was found that the cementite adjoining temper carbon areas was usually much smaller than that farthest away. Usually the area directly about the temper carbon was free from cementite. This would be the result if the carbide in the austenite decomposed. Owing to the liberation of the ferrite at the temper carbon the concentration of this consituent would be greatest there and the excess cementite in the immediate vicinity would be more quickly dissolved than that farther away.

The stability of the excess cementite in the white cast iron is shown again in Fig. 11. This specimen was annealed in a case-carbonizing mixture and the exterior portion retained the greater part of the excess cementite. Temper carbon and ferrite were also present in this rim. The interior showed a good malleable structure. In this case it is probable that the carbon necessary to replenish the austenite due to the decomposition of the dissolved carbide was obtained from the carbonaceous gases of the case-carbonizing mixture instead of from the cementite areas. The temper carbon in this rim was not surrounded by ferrite, but was scattered throughout the cementite areas, showing that the latter was more resistant than the gases to the dissolving action of the ferrite liberated at the temper carbon.

The decomposition of the iron carbide in the austenite is not complete at the annealing temperature, but is probably complete at 700°. By cooling quickly from the annealing temperature the specimen consists of nearly pure pearlite and temper carbon, or, if a piece of good malleable, consisting of ferrite and temper carbon, is heated to the annealing temperature, part of the carbon is quickly redissolved to form pearlite. By then annealing at 700° all of the carbon may be reprecipitated.

This pearlite formation shows that the stability of the cementite dissolved in the austenite is a function of the temperature, which is determined more or less by the silicon and manganese ratio. The lower the temperature the less stable the iron carbide. The determination of the limiting temperature or where the cementite is completely unstable was not made, but is probably close to the eutectoid temperature, about 725° to 775°. By cooling slowly through this temperature range the iron carbide may be entirely decomposed and ferrite and temper carbon obtained.

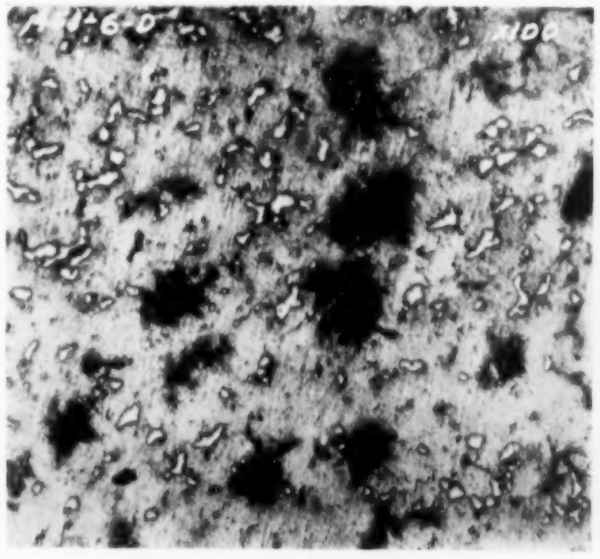


FIG. 8—MICROPHOTOGRAPH × 100

Since the cementite is unstable below this temperature the breakdown of the iron carbide may be possible below the Ar_1 point, but the reaction is slower since it is at a low temperature and the carbide is not in solution. Therefore, if the rate of cooling is slow until 650° or 700° is reached, the cooling may be more

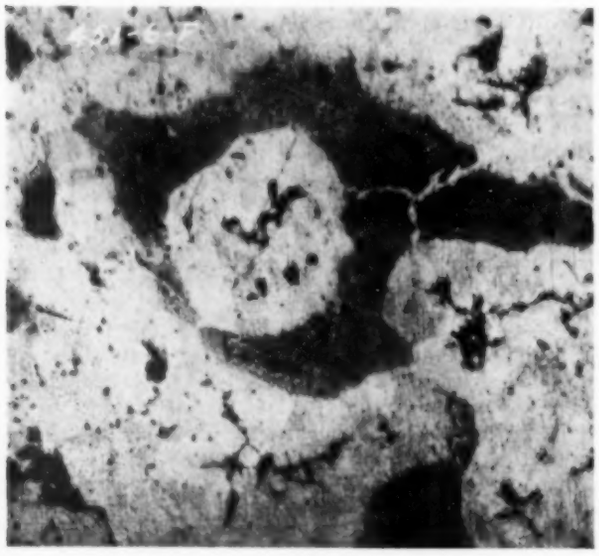


FIG. 9—MICROPHOTOGRAPH × 100

rapid to the room temperatures. This is corroborated by the experimental evidence.

Where the cooling from the annealing temperature is too fast the iron carbide forms pearlite. (See Figs. 9 and 10.) Since the austenite about the temper carbon particles is richer in ferrite the rejection of ferrite takes place here and the pearlite forms furthest from the carbon.

The structure of the rim is also dependent upon the rate of cooling. If the carbon is oxidized from the surface to an extent which will result in a medium to a high-carbon steel, as shown in Fig. 5, and the rate of cooling is slow enough to prevent any pearlitic formation in the interior of the specimen, the cementite in the rim will probably be stable owing to the absence of carbon nuclei. If the rate of cooling is too slow even this cementite will decompose forming a lower carbon steel or a pure iron skin, as shown in Fig. 6. Areas of such partly decomposed pearlite are shown in Fig. 12.

The skin effect has been a much discussed problem. Several different explanations of the disappearance of the carbon, or the decarbonization of the rim of malleable iron, have been offered. Some hold that the carbon migrates to the interior and remains in the casting, while others claim that the carbon migrates out and disappears entirely from the casting. It is obvious that if the carbon migrates out and leaves the iron at the temperature used for annealing, it must do so in the form of a gas. An analysis of the gases formed during the annealing would indicate their nature.

In the hardening and annealing of steels care must be used so that the surface does not become decarbonized. Experience shows how easily and quickly the surface carbon is oxidized. This surface decarbonization is not alone a result of the action of oxygen according to the following reaction:

$$2Fe_3C + O_2 = 6Fe + 2CO$$
$$2CO + O_2 = 2CO_2 \text{ (with excess oxygen).}$$

The surface reaction may also be the result of the action of CO_2 on the iron carbide.

$$Fe_3C + CO_2 = 3Fe + 2CO$$
$$2CO + O_2 = 2CO_2$$
$$Fe_3C + CO_2 = 3Fe + 2CO, \text{ etc.}$$

The amount of decarbonization is therefore dependent upon the presence of oxygen and carbon dioxide.

This reaction may be applied to the annealing process. Since the castings are somewhat inaccessible to the furnace gases the reactions and the rim decarbonization are slow. Where the furnace gases have a large excess of oxygen the reaction is more rapid than where less is used. The slower the oxidation the more steely the rim.

Even with a carbon packing it would be impossible to obtain a steel rim at certain low temperatures owing to the equilibrium relations between CO_2 and CO. This equilibrium calls for more CO_2 at the lower temperatures, and for this reason a casting annealed at a lower temperature, 700–800°, would probably have a more decarbonized rim than one annealed at above 800°. Where a carbon monoxide atmosphere is maintained as in the case of the specimen annealed in a case-carbonizing mixture, no surface oxidation results.

Some experiments were run to determine the gases resulting from the annealing operation. The reducing nature of the gas was shown in a preliminary experiment where a specimen was annealed in cupric oxide. The oxide was reduced, resulting in a mass of metallic copper. Samples of gas, which were being given off by specimens, were then taken and analyzed.

The following gas was taken from a sample being annealed at 870°.

	Per Cent
Carbon dioxide	13.2
Hydrocarbons	0.0
Oxygen	8.8
Carbon monoxide	26.8
By difference, Nitrogen	51.2

A second sample was taken from one annealing at 980°.

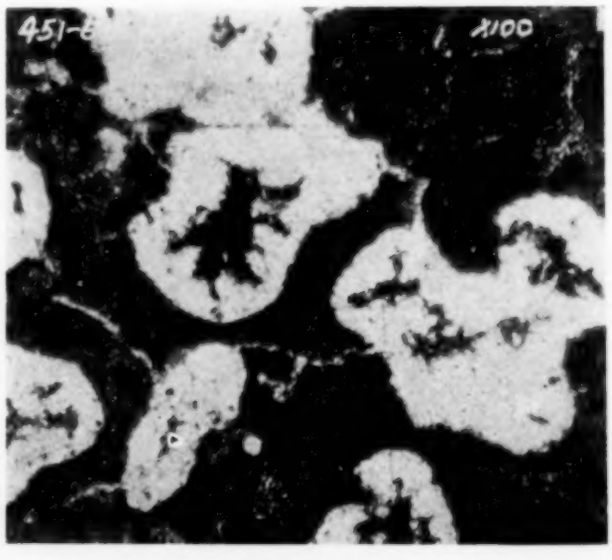


FIG. 10—MICROPHOTOGRAPH × 100

	Per Cent
Carbon dioxide	4.6
Hydrocarbons	0.0
Oxygen	3.4
Carbon monoxide	71.2
By difference, Nitrogen	20.2

The above analyses show that the carbon is removed from the casting as a gas, the composition of the gas depending upon the temperature and oxygen present.

General Summary and Conclusions

1. This research has resulted in the affirmation of some of the established laws in the annealing of malleable cast iron, namely:

(a) That the packing material does not affect the interior structure.

(b) Slow cooling is necessary to obtain good malleable.

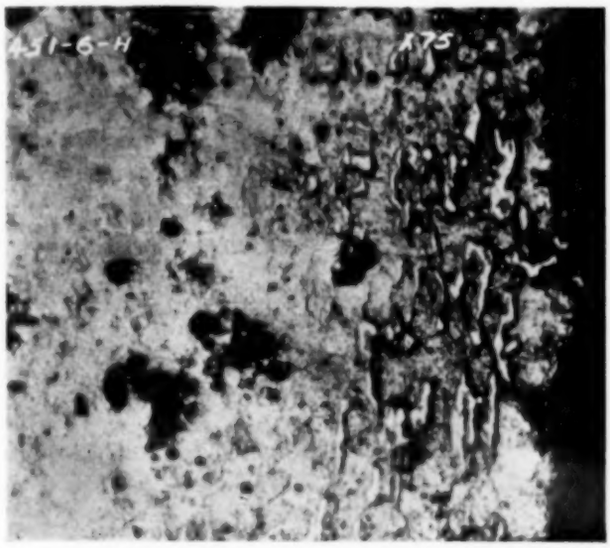


FIG. 11—MICROPHOTOGRAPH × 75

(c) A low temperature anneal gives a much better grade of product than the high-temperature malleable.

2. The conditions influencing the rim structure have been investigated.

3. The result of too rapid cooling has been studied, and is shown to result in a steely structure.

4. The breakdown of the iron carbide has been studied, and though present results are not final all seem to show that cementite does not break down

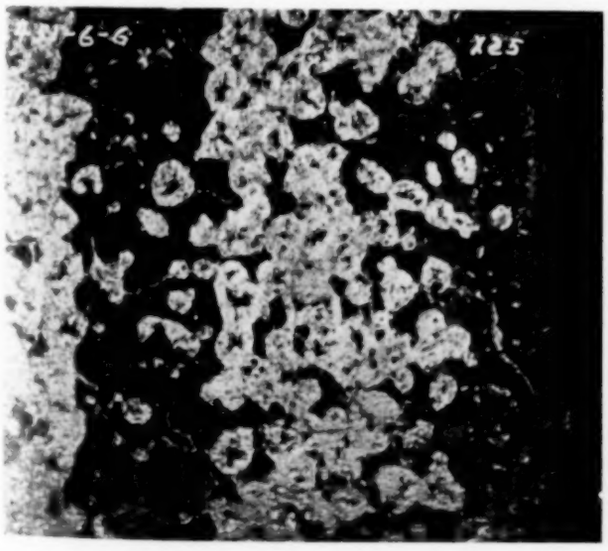


FIG. 12—MICROPHOTOGRAPH × 25

directly, but decomposes from the solid solution or austenitic condition. This breakdown may be compared to the electrolysis or decomposition of a salt. While it would be difficult to decompose a salt in the dry state it is a simple matter to cause decomposition when dissolved in water. The ionic theory calls for a dissociation into ions when such a salt is dissolved in water. Cannot the case of the decomposition of iron carbide be compared to the decomposition or dissociation of a salt? The analogy is not perfect, but a comparison can be made.

5. The decomposition of the iron carbide is complete above the Ar_1 point of the iron-carbon diagram. Whether iron carbide is unstable below the Ar_1 point has not been determined.

6. The decomposition of the iron carbide is not complete at the annealing temperature, but is complete in a range of temperature just above the Ar_1 point, as given in No. 5.

The author wishes to express his appreciation of the thorough experimental work done by Mr. Leasman, and of the valuable assistance given by the Northern Chemical Engineering Laboratories in the preparation of this paper.

Chemical Engineering Laboratories,
University of Wisconsin.

Wheeling Creek Mine Explosion

Carbide Lamps Freed from Suspicion

Gas and dust was the cause of the explosion which resulted in the death of Frank Hora at the Wheeling Creek Mine of the Lorain Coal & Dock Company at 2 o'clock, January 27th, 1914.

It was at first supposed that the residue from carbide lamps had caused the explosion; some of this residue was found in break-through 5, about 150 ft. from the center of the explosion. As a preliminary, pending the investigation, a temporary order was issued forbidding the miners of Wheeling Creek Mine to use carbide lamps. The local union passed a resolution demanding a thorough investigation and refused to go to work unless allowed to continue using carbide lamps.

In the meantime Chief Mine Inspector Davies, Deputy Inspector Gaffney, together with Safety Commissioner of Mines Roan, of the State of Ohio, aided by other deputies, officials and experts, were making a most searching investigation. It was soon demonstrated that as the residue left by the carbide lamps is merely slaked lime it could not be considered as a cause and it was proved conclusively that even had there been fresh carbide in the break-through it could not have had anything to do with the explosion as the break-through was dry. Carbide will not give off sufficient gas when exposed to the air to cause an explosive mixture or an ignitable mixture unless brought in contact with actual water. The moisture in the air is not sufficient to make enough acetylene to ignite. There can never be an accumulation of gas produced from carbide by moist air for the obvious reason that in order to produce gas any air which has given up its moisture to produce acetylene must move on and the acetylene goes with it. The careful investigation of the location of the carbide residue showed that as a matter of fact, this residue was shut off from the butt entry in which the explosion occurred by a substantial brattice, the residue being located within 3 ft. of the adjoining butt entry in break-through 5, and, as before stated, with a brattice between it and the butt entry in which the explosion occurred.

When these facts became apparent a meeting of the local union was called. The order prohibiting the use of acetylene was rescinded by the chief mine inspector, his deputies and the Safety Commissioner. The investigation into the actual cause of the explosion continued. The work of the Ohio mine inspectors and the Safety Commissioner was supplemented by an investigation conducted by Mr. J. W. Paul of the U. S. Bureau of Mines. It was found that the explosion was caused by an accumulation of methane (fire damp) intensified by dust.

Determination of Carbon in Iron and Steel

Direct Method

BY W. D. BROWN.

The essentials for complete combustion are an initial temperature of the furnace of about 1800° F., the burning of the metal in nearly pure oxygen, and an unlimited supply of the latter during combustion. If the metal is pushed into the furnace and a slow stream of oxygen started, combustion will begin before all the air has been displaced and will be in a mixture of air and oxygen; possibly the combustion of the iron will be complete before the oxygen has completely displaced the air and the only work remaining for the oxygen is to displace the carbon dioxide in the tube. When the combustion begins, oxygen should be supplied as fast as the iron will combine with it, that the temperature may be raised to the highest point; about 3000° F. will be momentarily obtained and all the carbon will be oxidized. These requirements of starting combustion in oxygen and an unlimited oxygen supply have been met in the method described below. In addition the supply of oxygen has been made automatic.

Apparatus. The electric furnace contains a silica tube ⅞″ in diameter and 24″ long; in it ignited asbestos is loosely placed from the exit end to within 3″ of the middle of the furnace; this acts very well as a catalizer, and is necessary when burning pig iron, at least. Oxygen is obtained in 100 ft. cylinders on which is a pressure reducer and regulator which supplies oxygen at a pressure from 1-3 lbs. as desired. The regulator is connected to brass piping containing four tees and globe valves for four electric furnaces (more could be added as desired), thereby one tank is used for the four furnaces. Oxygen enters the silica tube through one hole of a two-hole stopper. In the other hole is a 12″ copper or glass tube, ¼″ outside diameter; on the end of the tube is a No. 00 single hole rubber stopper. Through this stopper and tube a plunger rod of No. 9 nichrome wire, 24″ long, to which is riveted a sheet of nickel about ½″ diameter, is inserted. This is used to push the boat into the hot part of the furnace after the oxygen has displaced the air and while the current is flowing, and can be made in any laboratory; the No. 00 stopper can very well be replaced by a small stuffing box, in which case the copper tubing is shorter. The oxygen being free from CO_2, no purification is necessary. Both ends of the silica tube are cooled by copper water jackets.

The exit end of the tube is connected by a rubber stopper to a glass stopcock, which is used to regulate the flow of gas. The stopcock is connected to a 5″ drying tube filled loosely with phosphoric anhydride. It has been found that a zinc tube is not necessary, as the sulphur trioxide fumes are caught by the asbestos and phosphoric anhydride.

The drying tube is connected by a rubber tubing, on which is a pinchcock, to a Vanier potash bottle. This bottle contains a solution of potassium hydroxide (purified by lime) of 100 grams per 100 c.c. water. The inner tube is filled with phosphoric anhydride and a covering of glass wool placed thereon. Filled in this manner, the bottle will last for 100 determinations. The part which fails first is the phosphoric anhydride; a solution will form in the bottom of the drying chamber and will gradually grow until there is no more dry anhydride; this marks the life of the bottle.

On the construction of the Vanier bottle depends the success of the method; it must be so made that a speed of 400 c.c. per minute may be obtained without the gases passing out the lower end of the inner coil, which can be done if the lower end points or slants downward instead of being horizontal. The exit end of the coil should also be opposite the entrance to the drying chamber to prevent the splashing of the potassium hydroxide solution into the phosphoric anhydride. (Such a bottle is supplied by A. H. Thomas Co.)

The Method. A blank is first run in the following manner. With all connections closed, the oxygen is turned on the line and the pressure varied until three lbs. are obtained. One or more bottles are then connected to the furnace or furnaces and the stopper inserted in the entrance of the silica tube, the stopcock in the bottle is then turned, the pinchcock opened, and the globe valves from the oxygen line opened full. Oxygen should then pass through the bottle at a rate of approximately 400 c.c. per minute, the pressure in the line dropping to 2 lbs. If the speed of the gas is not as great as this, the reducing valve should be so regulated as to increase the pressure. It may happen that one bottle will go faster than the other; in that case, the stopcock at the exit end of the furnace is to be partially closed so that all bottles have the same speed. When this condition has been obtained,—the right pressure in the oxygen line and the stopcocks just right, no change is made until it is necessary to renew the bottle or tank. The stream of oxygen is continued for five minutes, the oxygen is then shut off at the globe valves on the oxygen line and, as soon as the flow of gas slackens in the bottle, the pinchcock on the rubber tubing leading to bottle is closed and, when the flow has completely stopped in the bottle, the stopcock is turned, the bottle is removed and weighed, preferably with a tare. This is repeated until the variations between weights is not more than three decimiligrams.

For the actual test, 1.36 grams of steel between 20 and 60 mesh or half gram iron, are weighed and transferred to a groove in alundum contained in a nickel boat. The bottle is connected to the furnace as above and the stopper turned, the pinchcock being closed. The boat is placed just within the silica tube, the two-hole stopper carrying the pusher is then inserted in the tube, the pinchcock opened and the oxygen turned on full at the globe valve. Oxygen then passes through at the rate of approximately 400 c.c. per minute; if the rate varies perceptibly from this it can be regulated by stopcocks at exit of silica tube. After gas has passed through for one minute, the boat containing the drillings is pushed into the center of the furnace by shoving in the nichrome wire mentioned before, while the current of gas continues. In about one minute the drillings have become heated sufficient to burn freely and combustion then is very rapid, being complete in twenty seconds. During the combustion an unlimited supply of oxygen is at hand, as the globe valve is open full and the regulator is set for a definite pressure. The oxygen then passes through to sweep out the carbon dioxide formed, five minutes after the boat is pushed back being sufficient. The globe valve is then shut off and, when the flow of gas has slackened, the pinchcock is closed, followed by the closing of the stopcock as above described. The bottle is disconnected and weighed; increase in weight in grams, multiplied by 20 gives %C. when 1.36 grams steel has been used.

In routine work, one analyst weighs up four (or less) tests, places boats in furnaces and starts the gas flowing. During the one minute's wait, he examines stoppers, etc., for leaks, then pushes boats back and leaves furnaces for other work, returning in five minutes for the completion of the determination. This method can be used for the determination of carbon in steels ranging from 0.07 per cent to 1 per cent carbon, as well as on pig iron. Not only is the accuracy greater than with color determinations, but the time required for analysis by combustion is less than by color.

Duquesne, Pa.

A Convenient Method for Handling Chemical Solutions in Quantity

BY C. C. PHELPS, M.E.

When acids and other chemical solutions of a corrosive nature must be lifted or conveyed from one place to another, the relative advantages of different methods of accomplishing the desired result cannot be decided on the questions of initial or operating costs alone, for the service may be so severe on certain types of apparatus as to render them practically useless in a short time. Frequent delays for repairing may also occasion great losses due to the stoppage of manufacturing processes. Ordinary reciprocating and rotary pumps are often out of the question, even when constructed of special material of a non-corrodible nature.

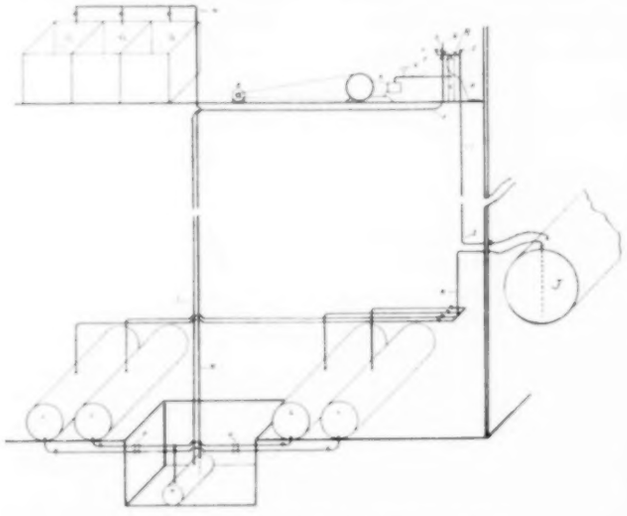


FIG. 1—SYSTEM FOR PUMPING ACIDS

A device known as a montejus, which is operated by either air or steam pressure, has been used to some extent for this purpose and is fairly satisfactory; but it is open to the objection that it has valves, floats and other moving parts in contact with the fluid. Other devices, using air or steam pressure, employ either the injector or siphon principle for their operation with more or less success, depending upon the requirements, but they, too, have certain limitations in practical use.

For handling chemical fluids in large quantities the method illustrated herewith has certain advantages which will be apparent from the description. The system illustrated in Fig. 1 is in use in a large Massachusetts textile mill, for raising heavy commercial oil of vitriol to bleaching vats on the third floor of a building. Compressed air is used, not only for this purpose but also to empty the acid from the tank cars which are brought alongside of the building. The engineers in charge of this plant consider that the method which they have adopted is admirably adapted for its purpose.

Referring to the diagram, O_1, O_2, O_3 are lead-lined bleaching vats; they are partially filled with pure water, after which 100 gal. of the concentrated sulphuric acid is raised and added slowly to the water, the contents being stirred while the acid is being added; then a small quantity of hydrochloric acid is added by hand from a carboy, after which the bleaching solution is ready for use.

Located near the vats is an air compressor *A* driven by a 2½-hp belted motor *B*. The compressor cylinder has a displacement of 15 cu. ft. of free air per minute and operates normally at 40 lb. per square inch air pressure. The compressor, gauge, receiver and controlling valves are shown in the photographic view, Fig. 2.

The compressor discharges into a length of pipe *C* of large diameter, which acts as a receiver to steady the pressure in the line. Gauge *Q* indicates the air pressure in the line. Two air lines, *I* and *F* lead from the receiver. By closing all valves on the left of the receiver outlet and opening valve *R*, pressure is put on pipe line *I*. The latter line leads to a connection whereby air pressure can be put on the tank of an acid car *J*. Safety valve *E* is set for 15 lb. per square inch, so no pressure in excess of this amount can be put on the car tank. The car tank's outlet is coupled with pipe *K* leading from the outside of the building to four heavy iron reservoir tanks *L* in the basement of the mill. Each of the tanks *L* has a capacity of 3,500 gal. and is provided with air vents. With an air pressure of 15 lb., it requires about two hours to empty the car, capacity of which is usually about 5,000 gal.

Pipes lead from the bottom of the reservoir tanks to the top of a 100-gal. iron supply tank *M* placed a few feet below. In each of these connecting pipes is a check valve *P* to prevent the acid from returning to the

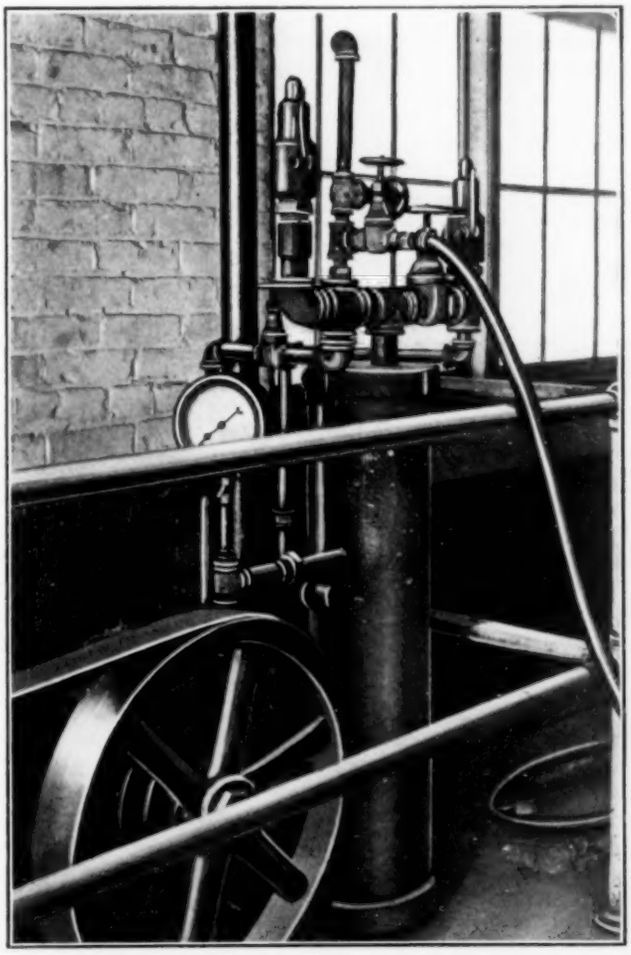


FIG. 2—COMPRESSOR RECEIVER AND CONTROLLING VALVES

large tanks. Leading from the top of the small tank is the air pipe *F*. Normally this serves as a vent, allowing the acid to flow into the small tank and fill it. When a charge of acid is wanted, this vent is closed by a valve *S* near its outlet *G* and air pressure is put on this line by opening valve *T*. The acid immediately begins to flow from the bottom of the 100-gal. tank up pipe *N* to the bleaching vat, a height of about 40 feet.

About five minutes is required ordinarily to pump the 100 gal. wtih an aair pressure of 40 lb. Saafety valve *D* is set for 50 lb. per square inch. A short length of hose, *H*, connected to the air system, is used for cleaning and similar purposes.

With this system there is no mechanism to come in contact with the acid and get out of order. Everything is simple. When a charge of acid is wanted, the motor is started by simply throwing over the starting switch; then the proper valves are opened. The charging tank will not deliver more than a hundred gallons at a time, although less than that quantity could easily be obtained by shutting the valve at the vat at the proper time; the air pressure on the tank being constant, the rate of acid flow will, likewise, be constant, and the quantity of acid lifted in a given time will, therefore, be proportional to the time required for raising it. Pressure is put on the line only when actually pumping, hence the economy of operation is marked. The air pressure is on only the small tank and the lines leading to and from it. The larger tanks are never under air pressure and hence any trouble that might develop through leakage in such large tanks is avoided. The acid, being in concentrated form, does not cause any trouble from corrosion in the iron tanks or pipes, as might occur with dilute acid.

The flow of acid by gravity from the reservoirs to the small tank is entirely automatic and reversal cannot occur. No attention is given to the apparatus in the basement, the entire control being where the acid is used. The extreme simplicity of the arrangement described above ought to suggest its use for a large number of industrial processes, perhaps with modifications to suit each case.

Weehawken, N. J.

Sixth International Congress of Mining, Metallurgy, Engineering and Economic Geology

London, 1915

The Sixth International Congress of Mining, Metallurgy, Engineering and Economic Geology will be held in London from Monday, July 12, to Saturday, July 17, 1915.

An Executive Committee has been formed with Arthur Cooper, LL.D., as chairman; Sir William E. Garforth, LL.D., as vice-chairman; Sir Hugh Bell, Bart., D. C.L., LL.D., as honorary treasurer. Mr. G. C. Lloyd is the general secretary with offices at 28 Victoria street, London, S. W.

There will be four sections, section 1 being devoted to mining (coal mining and metalliferous mining); section 2 to metallurgy (chemical metallurgy and physical metallurgy); section 3 to engineering; and section 4 to economic geology.

Members of the Congress may be:

(a) Honorary members and the official delegates of foreign states.

(b) Supporters of the Congress—that is, members who subscribe not less than £5 ($25) to the Congress fund.

(The above will be entitled to attend all meetings and to receive all publications of the Congress.)

(c) Ordinary members, who pay a Congress fee of £1 ($5), which entitles them to registration in any one of the four sections, to attend all meetings, and to receive the Proceedings of that section in which they may elect to register. They may register their names in, and obtain the Proceedings of the other Sections by making a further paying of 5s. ($1.25) for each section.

The Congress fee for ladies is 16s. ($4).

The Development of Modern Cyanide Practice

BY C. M. EYE., E.M.

In reviewing the rapid advance of the cyanide process for the recovery of the precious metals, one is most impressed by the broadened field of its application to-day as compared with that of the early days of the process. Many of us can remember when its successful application was limited to clean, porous, well-oxidized gold ores, the treatment being by leaching. The primitive method consisted of crushing dry to as coarse a mesh as possible, usually ¼-in. size or coarser, charging dry into shallow tanks with no attempt at separation of fines, and then leaching. The first wetting was by solution put on from below, followed by downward flow of solution, closely followed after the proper period by weak solution, and in this turn by water wash. The principal precaution taken was to avoid the separation of fines or slimes into a layer or layers at any stage of the treatment. The writer's first experience was obtained in a plant of this character, where, in spite of the presence of considerable clay in the ore, better than 90 per cent extraction was obtained regularly.

Following such direct treatment, the next logical development was its application to the tailings from other processes, notably amalgamation, and in this a much greater variation of practice developed. The treatment generally involved double handling, in order to rid the tailings of excess moisture, break up slime layers and render them permeable. In those days, since separate slime treatment was expensive and unsatisfactory, as large a proportion of slime as possible was handled with the sands by leaching. This rendered double handling almost unavoidable. The results on sand were poorer than if the slime had been kept out, but it was a case of choosing the lesser of two evils.

In many cases, however, direct filling and drainage was practiced, with poorer results, it is true, but at less expense. More slime had to be eliminated in this case; if not by previous classification, then in arrangements for securing overflow of slime and excess water as the tanks filled, by superimposed rings in the center as in South African practice, or by plugs and drains in the sides. Of course, the moisture content could not be reduced to as low a figure by direct filling, nor was the charge left as pervious for leaching, being packed after draining. Another difficulty encountered in this method of treatment, more pronounced than in others, was that of aeration to secure the oxygen necessary for extraction. Frequently preliminary treatment was followed by transfer and additional leaching in order to obtain additional oxidation and aeration. Attempts were made to introduce air from the bottom, but these usually resulted in channeling. About every known oxidizing agent was used, first and last, to artificially supply this necessary element, but were usually abandoned because of cost, inefficiency or deleterious effect on solution. The practice of using artificial oxidizing agents has therefore practically gone out, through aeration of solutions now fulfilling this function.

Steps in Mechanical Development

The mechanics of the earlier developments were simple, the greatest development in connection with leaching methods being reached by the Blaisdell system of filling and emptying tanks by an excavator on the principle of a disk harrow, but this system was and is applicable only to large installations on account of its cost. The revolving sand filter for the preparation of sand for leaching is another extremely important development. These, with the simple but efficient But-

ters distributor, comprised the more important mechanical developments, until the invention of the submerged slime filter. This opened up an entirely new field for expansion, for it not only furnished a method of separating solution from slime much superior to the decantation method, until then followed for the treatment of slime unavoidably produced, but it led to the general adoption of fine grinding of sand, and the treatment of both together by agitation and vacuum filtration. The multiplicity of patented mechanical devices and methods using this principle led to so much litigation, however, that operators came to regard them all with mingled feelings of uncertainty and disgust, and came to regard vacuum filtration as something to be used when necessary, but to be avoided where possible. It was found also in many cases in practice that the net results from the use of filters were not so great as at first supposed on account of difficulties and expense of operation and upkeep.

Not that vacuum filtration has passed, or is likely to be relegated, by any means, for in many cases conditions are such as to make this the only practicable method of working; but the introduction of continuous treatment has made it possible to dispense with filtration by using other and more simple systems of separation. Foremost in this has been the continuous countercurrent decantation system, using thickeners for obtaining a low ratio of solution to solids, and involving the handling of solution instead of pulp with less power and less expense. The principal objection to this system is that while it affords an ideal means for very complete recovery of gold, it does not admit of as close recovery of cyanide solution without undue building up of stock solution, and for this reason has frequently to be followed by vacuum filtration to avoid heavy mechanical loss of cyanide.

Upward Displacement in Slime Treatment

A system which rivals this and which is gaining ground, is based on the principle of upward replacement, the barren solution or wash being introduced at the bottom and displacing the solution which it is desired to remove. At first sight it would appear that diffusion would defeat the object sought, but when the displacing liquid is mixed with a thickened pulp at the bottom of a tank (this pulp being constantly added to by settling pulp from above and removed by drawing off constantly from the bottom), the displacement does take place with very little diffusion. It is probable that the proximity of the individual particles of the pulp in a thickened but thoroughly uniform mass, prevents the diffusive action which otherwise would occur. It is noticeable, also, that the diffusion is more pronounced between solutions of different chemical strength, and greatest between clear water and solution, as would be expected. While open to the objection that the use of slime-circulating pumps is essential, the system has several decided advantages: First, that in the replacement of gold-bearing solution, very little dilution of the solution takes place, enabling the metals to be recovered from a smaller bulk. Second, that only a small proportion of the gold is carried forward to the next stage with the pulp, as the solution going with the pulp is mostly barren solution that has been added. Third, that it affords a means of changing from solution of one chemical strength to another, or from water to solution or vice versa, with very little building up of stock or loss of chemicals. This latter point makes for the system two unique uses: First, in displacing water by solution from pulp where the preceding mill treatment is conducted in water, and, second, after cyanide treatment, in displacing solution by water for final discharge.

The extensive adoption of the practice of crushing and grinding in solution, dispensing with the amalgamation or other treatment, has obviated in many cases the necessity of close separation of liquid and solid befor the latter enters the cyanide plant, a thickening to the desired degree of dilution for agitation being all that is required. It is here that the continuous countercurrent decantation system has its best application, it being possible to operate the whole plant with one closed solution circuit. But it is not always practicable to crush and grind in solution, owing to possible necessity of amalgamation to recover coarse gold not easily dissolved in cyanide, or necessity of concentrating or otherwise removing from the ore such constituents as would be destructive to cyanide in direct treatment. With these conditions, it becomes essential to remove as much liquid as possible from the pulp before it enters the cyanide plant, to avoid undue accumulation of solution and possible destruction of cyanide by soluble cyanides. It has even been found advisable to use vacuum filtration at this point, reducing the moisture to about 30 per cent, as against a usual 50 per cent by ordinary thickening. Even the 30 per cent remaining which has to be made up into cyanide solution of full working strength in the treatment, may cause a considerable accumulation of solution, and if the principle of upward replacement can be successfully applied here, with a resultant saving in cyanide, it will be a distinct step in advance. It is claimed by those introducing it that it is being so applied, but few data are available as yet as to this use. It is to be borne in mind that thickening is done at the same time, and in the same operation, though somewhat more expensively than ordinary thickening because of the necessity for operating a circulating pump in connection.

Present Tendency in Agitation

In agitation methods, compressed air is the almost universal agent employed, both on account of its convenience of application, its value in aeration, and its cost, as compared with mechanical agitation, when upkeep of the latter is taken into account. Its use in air lifts as a means of pulp transfer is almost as nearly universal, and for the same reasons. The Pachuca is still the most popular air agitator, being in most cases now operated continuously in a series of several tanks. The extreme height at first thought necessary has been dispensed with in more recent installations, enabling these tanks to be used with less air pressure and equally good results.

Systems using circulating slime pumps are no longer in general use, owing principally to difficulty of keeping up the pumps, and the cost of operation. Modern slime pumps are being built, however, much superior to those of earlier days, with well protected bearings; and when run at proper speeds, and well cared for, with stuffing boxes protected at all times by clear water or solution under pressure, there is no good reason why they cannot be maintained. They allow of one important feature in treatment which is very beneficial, and that is the introduction of atmospheric air in abundant quantity, intimately mixed with the pulp, this air being free from the hydrocarbon compounds resulting from decomposition of lubricants in compression.

Probably the latest successful agitator to be placed on the market combines the mechanical agitator with the air-lift principle of the Pachuca. In fact, several machines have been devised lately on these lines, but with different methods of application. This method may be regarded as the present tendency in agitator construction.

In precipitation, the old, original zinc shaving method of extraction is still used generally in smaller

installations and out-of-the-way places, on account of its simplicity; but the perfection of the zinc-dust method, its cleanliness and ease of operation when properly installed, and its better adaptability with solutions weak in cyanide has led to its adoption in the majority of new and larger installations. The newly developed method of precipitation by the use of aluminium is no doubt the most nearly perfect method yet devised. Cleanup and refining methods vary much with the practice in precipitation, but the washing and direct smelting of carefully screened precipitates in oil-fired tilting furnaces covers the majority of cases.

The extension of cyaniding to ores of gold and silver combined, then to straight silver-bearing ores, some of which are very complex, is perhaps the most wonderful feature of the development of the process. Facilities for fine grinding and thorough agitation with air were the principal factors in this. The record made in scores of modern mills makes the chronicle not many years since of a 60 per cent extraction at Chloride, Utah, on ore containing chloride of silver, seem like a tale of the Middle Ages. And that was a record in silver cyanide at the time. Indeed, it is hard to realize how this process has, in twenty-five years or so, displaced to a large extent amalgamation (becoming its chief assistant where it has not displaced it), relegated to the past chlorination and silver amalgamation, and reached out into every part of the globe until it is to-day the most generally recognized and applied of all wet metallurgical processes.

Los Angeles, Cal.

Development of Alaska

The final passing by Congress and signing by the President of the Alaska railroad bill is of interest to the mining and metallurgical industry. The development of a country acknowledged to be rich in mineral resources has long been delayed due to the inability of interested parties and the government to agree on a system under which the public lands should be exploited. Coal lands in particular have been the bone of contention. It is expected that the railroad bill will now be followed by one providing for leasing of coal lands. The provisions may not be all that could be desired, but in the main they are favorable. Comparatively large acreages, 2650 acres, are to be leased, and all the coal therein contained is covered by the lease. Royalties are regulated, above a minimum of two cents per ton, by competitive bidding among the lessees. With the development of coal lands in Alaska there should come an increased interest in its other mineral resources, and the mining industry should develop largely.

The British Aluminium Company

The fourth ordinary general meeting of the reconstructed British Aluminium Co., Ltd., took place on March 27th, when Mr. A. W. Tait, chairman of the company, presided. He said that the trading profits for the year and the proceeds of investments, together with the revenue from the Kinlochleven & Foyers Estates, amounted to £268,720 5s. 2d., as against £194,823 15s. 11d. in the previous year; and these results are by a long way the best ever accomplished by the company since its original formation in 1894. The increase in the net profits is entirely due to trading and is attributable to increase of tonnage, economy in cost of production and, to a still greater extent, to the higher prices prevailing during the year. These prices, however, had not been abnormal, but have only returned to a reasonable figure after four or five years' depression.

The revenue from the Kinlochleven & Foyers Estates shows a decrease of £906 6s. 8d., as compared with the previous year, which is almost wholly due to unusual expenditure connected with the village at Kinlochleven, which was necessary for the housing of the workmen. Additional plant has been put down at the Aluminium Works at Vigelands, Norway, from which an increased production for the present year is expected.

The large increase in the productive capacity of the company's aluminium works has made it necessary to correspondingly increase the capacity of the works for producing alumina. Last year the company acquired a half-interest in the works of the Bauxite Refining Company at Hebburn-on-Tyne, but the increase so obtained is not now sufficient to meet present or future requirements, and arrangements have been completed to acquire about 38 acres at Burntisland, Scotland, where the conditions of labor supply and shipping facilities are favorable. To ensure a good supply of bauxite a quantity was acquired from the Union des Bauxites and has been deposited on land in the name of the company. This will form a reserve stock, only to be used to supplement regular supplies from the subsidiary company. The company's rolling mills at Milton, Staffordshire, have been employed up to their full capacity during the whole year. The new mills at Warrington, Lancashire, have been completed, and work was partially begun there at the commencement of this month.

With regard to the accounts, the depreciation reserve had been increased from £100,000 to £150,000, and the reserve account from £80,000 to £130,000. The profit on the year's working, after making provision for the prior lien debentures and debenture stock, and making the requisite allotments to depreciation and reserve accounts, justified the directors in recommending payment of the full dividend on the preference shares and 5 per cent on the ordinary shares. This was the first time the company had paid a dividend on its ordinary shares since the reconstruction.

The report was unanimously adopted.

Scotch Ship-Building

Although it is unquestionable that there is a considerable decline in the ship-building trade, yet the orders placed during 1913 will ensure that the depression will not be felt in Scotch yards for some time yet; and at present the Clyde output is quite satisfactory. The total Scotch launchings for January amounted to 19 keels of 30,812 tons, of which 11 ships totalling 22,272 tons were Clyde-built. In January of last year only five vessels with a total of 9,755 tons left the stocks on the Clyde, and although the output for this January does not include any large liners or naval vessels the general average for the month has been maintained.

Two ships launched from Scottish yards early in February are of unusual interest to marine engineers. One, the Mississippi, built by Messrs. Harland and Wolff at their Glasgow yard for the Atlantic Transport Line, is a twin-screw vessel of some 5,000 tons gross, driven by internal combustion engines of the Diesel type made by the Burmeister and Wain Oil Engine Co., of Glasgow, and is the largest ship of the kind hitherto built on the Clyde, although she is the first Messrs. Harland and Wolff have built for internal combustion engines.

The other ship is the Transvaal, of 4,383 tons gross, built by Messrs. Ramage and Ferguson, of Leith, for the East Asiatic Co., of Copenhagen, and is the largest steam vessel yet constructed at Leith.

Blast Furnace Stoves

BY J. E. JOHNSON, JR.

In the third decade of the nineteenth century James Beaumont Neilson, who from plain beginnings had made for himself a distinguished position as a gas engineer and authority on matters of combustion, was consulted by a furnaceman as to the reason for the very unsatisfactory work of his furnace, particularly in Summer.

This started Neilson on a long investigation. Among other things he experimented with the use of preheated air on gas burners and on blacksmiths' fires and found that the effect was a marked increase in the brilliancy and intensity of the combustion. In 1828 he took out a patent on the use of preheated air in the blast furnace, but even after surrendering two-thirds of his patent to furnace owners he had the greatest difficulty in obtaining a chance to make a trial of the process.

Finally after several years of effort and after overcoming the most factitious and most unreasonable objections to changes in insignificant details such as the shape of the blast pipe he made a demonstration and while the increase in blast temperature was only 80 deg. Fahr. the results were so obvious that the system was at once adopted and higher temperatures obtained.

Then he was at once thrown into another difficulty—that of preventing the overthrow of his patent by a combination of Scotch furnace owners who took a pledge to break his patent virtually "by legal means or otherwise" in order to avoid the payment of his moderate royalty of 1s. per ton, though the saving effected amounted to 32s. per ton.

It is rather a relief to realize that such piracy, which we have been taught to regard as the product of our own times, flourished even in those "good old days."

It is still more pleasant to know that after innumerable trials which went thrice to the House of Lords, Neilson and his associates won, and that on the royalties he finally received he was enabled to pass the closing years of his life in comfort and leisure devoted very largely to what would now be called welfare work. The results obtained were probably very much beyond Neilson's most sanguine expectations, and within a few years the warm blast was used in producing by far the largest proportion of all the iron made. The reason for this saving so disproportionate to the amount of heat introduced by the warm blast will be explained in an article on thermal action of the furnace. We have here only to deal with the apparatus used for heating the air.

The first apparatus used was in all cases a nest of pipes of some sort in a masonry chamber, within which the gas from the furnace burned around the pipes while the blast on its way to the furnace passed through them. In early days these stoves were set on the top of the furnace, or if not immediately on top of it at the level of the top and just to one side, as has briefly been described in connection with boilers, so that the burning gas from the open furnace top could be drawn into them. This had the disadvantage that the cold blast had to be taken to the top of the furnace and the hot blast brought back to the bottom with the loss of heat consequent upon that journey, and as soon as the closed top was introduced the stoves were set at the ground level, with great advantages in first cost and in convenience.

The temperature to which the blast could be heated in these stoves was strictly limited by the fact that the pipes constituting the heating surface were rapidly destroyed if a certain temperature, found by experience, was exceeded by even a small margin, and if the stoves were fired while no blast was passing through them to cool them off, they might be completely destroyed in a period of an hour or two.

To meet this condition the late Sir William Whitwell conceived the idea of applying to this service the regenerator previously invented by Sir William Siemens, which consisted of a pair of chambers filled with brick work, through which passed a great number of passage ways arranged to present as much surface as possible.

Through these passage ways of one chamber the hot waste gases of combustion were passed for an interval sufficient to heat the bricks to a high temperature; then the gas was cut off from this and thrown into the second chamber, and the air to support combustion was drawn through the first chamber, from which it absorbed heat by direct contact with the hot surfaces of the bricks. When a sufficient amount of air had passed to lower the temperature of the bricks below that which would impart the required temperature to the air, the gas currents were again reversed in relation to the chambers, the gas being put back through the first one and the air currents being drawn through the second.

For melting the crucible steel, the first purpose to which Sir William Siemens applied his invention, the hot air for combustion was drawn through the checker work under the action of the stack draft, the pressure within the checkers, therefore, differed by only a few inches of water from that of the atmosphere, and masonry construction throughout was sufficient for tightness under this difference of pressure.

In order to apply this system to the blast furnace, however, the chambers had to be so constructed as to withstand the pressure of the air after compression to the pressure required for the blast furnace, because obviously the blowing engines could not handle blast heated to a temperature of 1000 deg. or so. Therefore, the blast must be compressed first, and after heating must go directly to the furnace.

For this reason Sir William Whitwell built his stoves in the form of cylindrical chambers with a top either cylindrical or dome-shaped to prevent deformation under pressure. The bottom was flat because prevented from deforming by the weight of brick work upon it. The shell of these chambers was of iron (later steel) plate, riveted together air-tight and strong enough to resist the maximum blast-pressure expected.

There was also another change in the conditions as compared with those of the steel melting furnace. There the gas to do the heating had already been burned and was simply hot waste gas. The waste gases of the blast furnace, on the other hand, are not particularly hot, but are highly combustible, and in order to provide a gas burner and combustion chamber these fire brick stoves required, therefore, four connections. First an opening for the gas burner, second an opening to the draft stack, third a connection for the cold blast from the blowing engine, fourth a connection for the hot blast to pass to the tuyères of the furnace.

All of these openings were required to be alternately closed air-tight and opened at frequent intervals, and must retain their tightness under conditions of temperature injurious to any metal available for construction. This meant that all metal parts around these openings had very largely to be water-cooled. The brick stove, therefore, brought many annoyances and difficulties of operation in its train, and, while it gives blast temperatures about 50 per cent higher than those permitted with the iron pipe stove, its first cost is greater, and this with the inconveniences and difficulties of operation and maintenance resulting from the conditions described above, retarded its development somewhat, so that a few plants have continued to use the iron pipe stove, but except in charcoal practice they rapidly disappear.

With this general view of the situation we may proceed to describe the two classes of stoves.

The Iron Pipe Stove

This stove had a development of about a half century before it was cut off by the preponderating advantages of the brick stove, and in this interval a type of construction was evolved which was almost universally used. This consisted of a large brick chamber lined with fire brick, beneath the brick floor of which passed horizontal pipes with nozzles or connections on their top surface, and a vertical partition in their central plane. On these were arranged U-shaped pipes reaching practically to the top of the chamber, through which the air had to pass on its way from the blowing engine to the chamber of the pipe at the opposite end is similarly prolonged into an outlet nozzle. Of course, these end chambers are without the lateral nozzles which connect the chambers to one another. The U-pipes are all of elliptical section in order to get a maximum of heating surface for a given area, and the nozzles on the bed pipes are similarly elliptical in section and of the same size as the end of the U-pipes, these ends being enlarged by a sort of collar cast on them which projects outside the body of the pipe about a half an inch, and which fills the spandrel between the two ellipses to a straight line, this being the shape of the exterior of the nozzles on the bed plates also.

It is perfectly obvious, on account of the central vertical partition in the bed pipes, that there is no passage

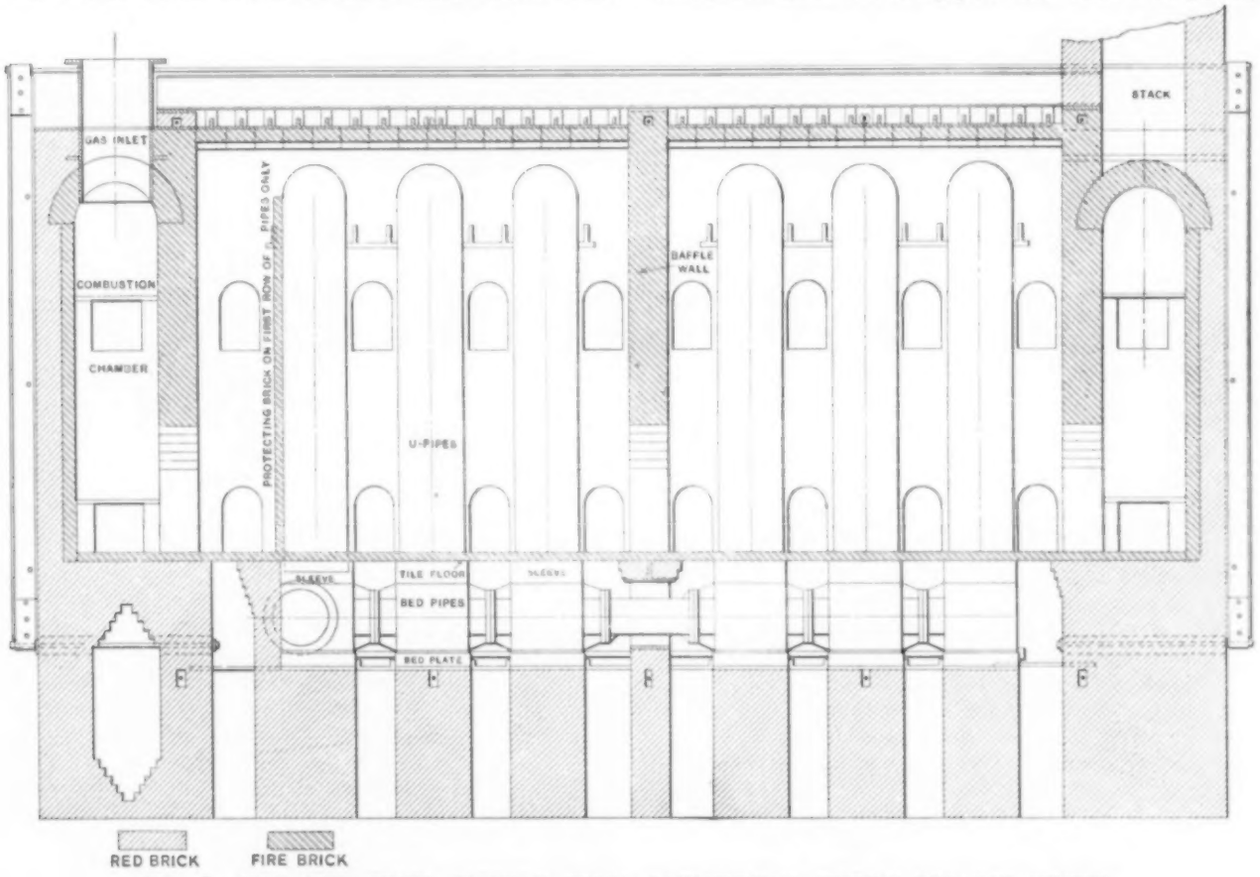


FIG. 1—IRON PIPE STOVE, GLENDON TYPE; LONGITUDINAL SECTION THROUGH CENTER

furnace; short-circuiting of the U-pipes was prevented by the central vertical partition. Around these pipes burned the furnace gas entering at one end and passing through the rows of pipes and through the baffle walls to the other end, where at last it passed out of the stack.

An excellent design of this type of stove is shown by Figs. 1, 2, 3 and 4, which illustrate a stove built by the Longdale Iron Company at their plant at Longdale, Virginia, under the designs of Mr. H. Firmstone, the president of the company. Fig. 1 is a vertical longitudinal section, Fig. 2 a vertical transverse section, Fig. 3 a sectional plan, Fig. 4 a detail of the U-pipes.

It will be seen that upon a masonry foundation rests a series of cast-iron bed-plates strongly bolted together and planed on their top surfaces. Upon these rest the bed pipes, square in cross section, with a central vertical partition and with three lateral nozzles on each side, in addition to the nozzles for the pipes on top.

None of these bed pipes have any connections on their ends except that the outer chamber of the bed pipe at one end is prolonged into an inlet nozzle, and the outer way for the air except through the U-pipes, and the bed pipes being joined in series, with the U-pipes in parallel, the air passes through an entire set of U-pipes on one bed pipe as a single unit, thence to the next bed pipe, and through all the U-pipes on that simultaneously, and so on.

THE JOINTS

The principle of these stoves is obviously very simple, but their construction and maintenance have developed a number of problems. The joints between the U-pipes and the bed pipes are universally sealed with what is known as a rust joint, a mixture of fine iron borings and sal ammoniac made into a stiff paste and caulked into place. This mixture rusts with great rapidity; this converts the whole into an homogeneous mass of iron oxide, which is firmly cemented to the adjacent surfaces of solid metal.

A cast-iron sleeve large enough to go over the nozzles on the bed pipes and the ends of the U-pipes, and about 10 in. in depth, so as to extend 5 in. on each side of the joint when resting on the bed pipe, is slipped

over each of these joints and the iron borings run down between it and the surfaces to be sealed. When this is properly done a first-class joint is secured. This is preserved as far as possible from the action of the heat of the direct flame by a floor of flat tile bricks laid between the rows of pipes and resting on the sleeves so as to make a fireproof brick floor above the level of these joints.

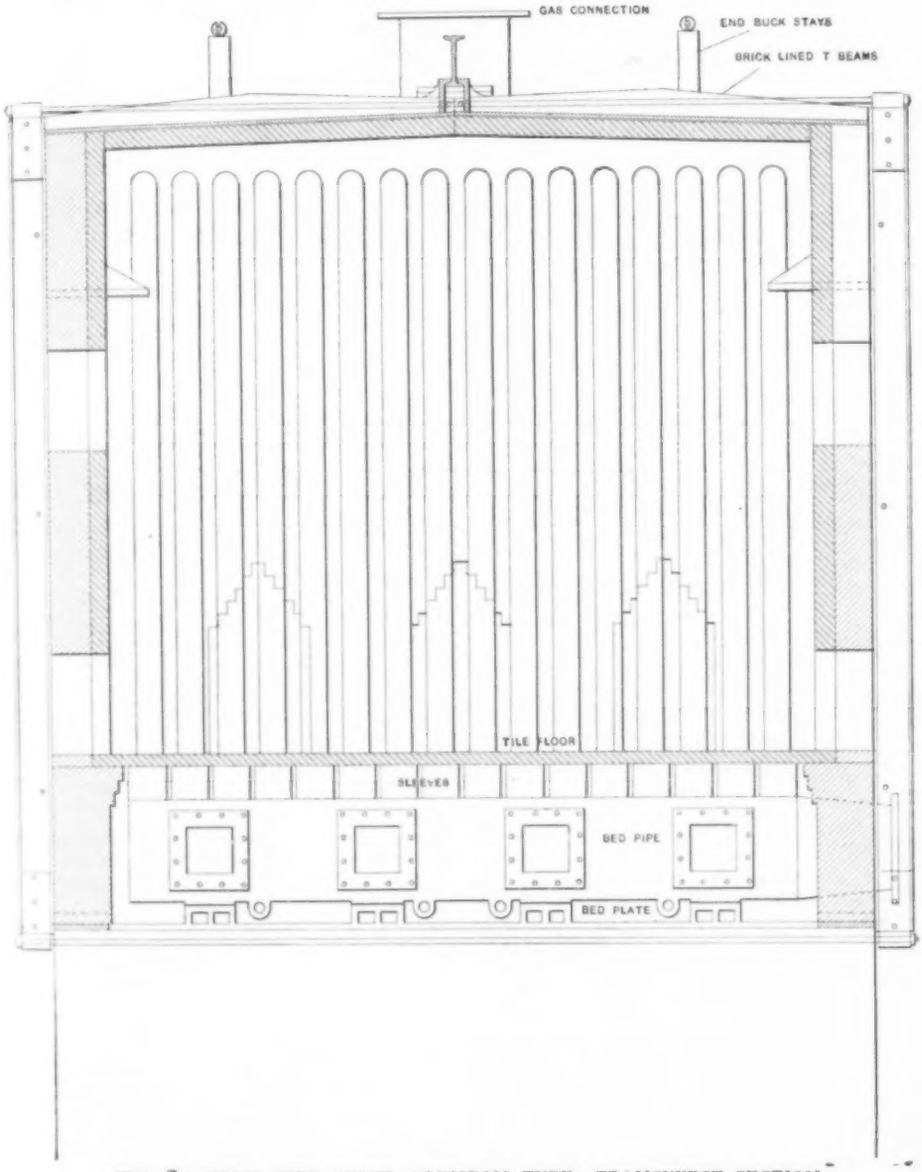


FIG. 2.—IRON PIPE STOVE, GLENDON TYPE; TRANSVERSE SECTION

The U-pipes immediately exposed to the gas coming in from the combustion chamber receive the severest treatment, and these are protected from the direct action of the flame by casting a dove-tail on the side of the pipes placed next to the combustion chamber and sliding over this dove-tail fire brick tiles with a similar dove-tailed slot. These are put on the pipes so as to cover them from top to bottom on the side next to the combustion chamber and greatly increase their life.

BAFFLE WALLS

In order to prevent the gas from the burners from passing along the top of the stove to the stack without going down around the pipes, baffle walls are built as shown, with openings through their bases through which all the gas has to pass. This forces the gas current to the bottom twice in its course to the stove and secures a good distribution of the flame over the surface of the pipes.

THE ROOF OF THE STOVE

Many stoves have been built with arched roofs, but these are a great inconvenience when U-pipes have to be taken out and new ones put in, as unfortunately happens not infrequently with this type of stove; and to tear out a portion of this fire-brick arch every time this requires to be done, would add considerably to the expense and delay caused by the operation.

The Longdale stoves were, therefore, covered by long cast-iron T-beams whose lower surfaces were of jaw-shaped or dove-tailed section. These were lined with tiles or shaped fire bricks, the upper part of which fitted into this jaw-shaped section, while the lower part projected out a little beyond the edges of the T-beam. These "shapes" were slid in from the end of the groove and laid in fire clay to make a tight joint and to hold them in place.

In small stoves one end of this T-beam rests on one wall of the oven and the other on the other, but in a large stove, such as that shown here, the span would have been too long for necessary strength, for convenience in handling, and for cheapness in casting the beams, therefore a steel I-beam runs down the center of the oven above the T-beams, and these were cast in two pieces, with the joint in the center, the end at the joint having a hook which rests on the lower flange of the I-beam. This excellent construction gives strength and complete protection of the iron work of the roof from the flame, together with convenience and accessibility. Moreover, this construction makes a level top for the stove, convenient to work upon.

This latter is important because when changing pipes shear poles of relatively short and convenient length are set up on top of the ovens resting upon the T-beams, and by this means can be placed exactly where desired, which could not possibly be done with an arched roof. By the use of this construction, where a U-pipe requires to be replaced the half-length T-beams immediately above it can be removed to make room for the operation and then restored to place after it was completed with a minimum of labor and time and without requiring work by masons.

THE SHAPE OF THE PIPES

The elliptical shape of the pipes is due to a desire to obtain maximum surface for a given area, which may seem at first sight unnecessary, because it might be

thought that the area was unobjectionable and that it would be simpler to make the pipes round and not bother about the increased area.

This, however, is open to two objections. First, the length of the bed pipe required for a given number of U-pipes would be greatly increased, which would be objectionable from every point of view, and second, velocity is a matter of much importance in securing proper transfer of heat, much greater transfer being effected at high velocities than at low. Therefore, by constricting the area the velocity of the air passing through the pipes is increased and the temperature obtainable with a given amount of heating surface thereby increased also.

Many stoves have been built with round pipes, and at the end next to the combustion chamber this is probably wise, because the relatively narrow cross-section of the elliptical pipes makes them bend about much worse under the influence of the flame than do circular pipes whose diameter is equal to the major diameter of the elliptical pipe. But in the rows which are further from the combustion chamber this does not hold to anything like the same extent, and there is probably ample justification for the use of the elliptical section in those rows.

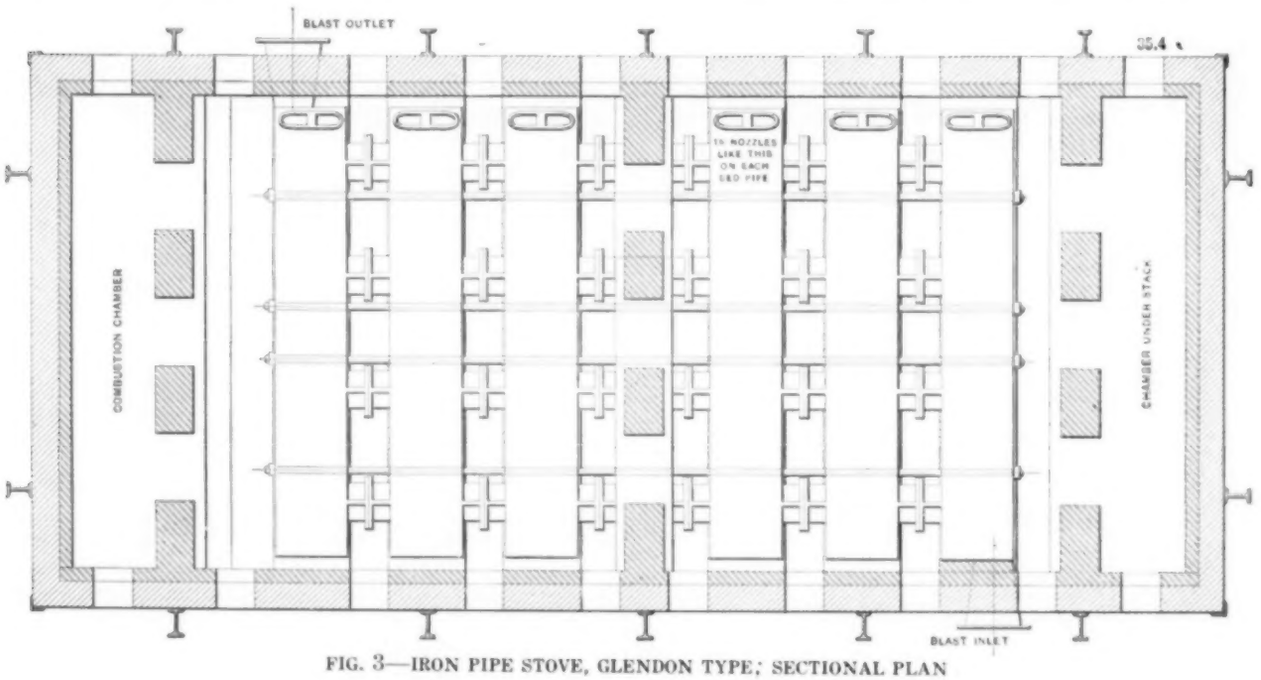


FIG. 3—IRON PIPE STOVE, GLENDON TYPE; SECTIONAL PLAN

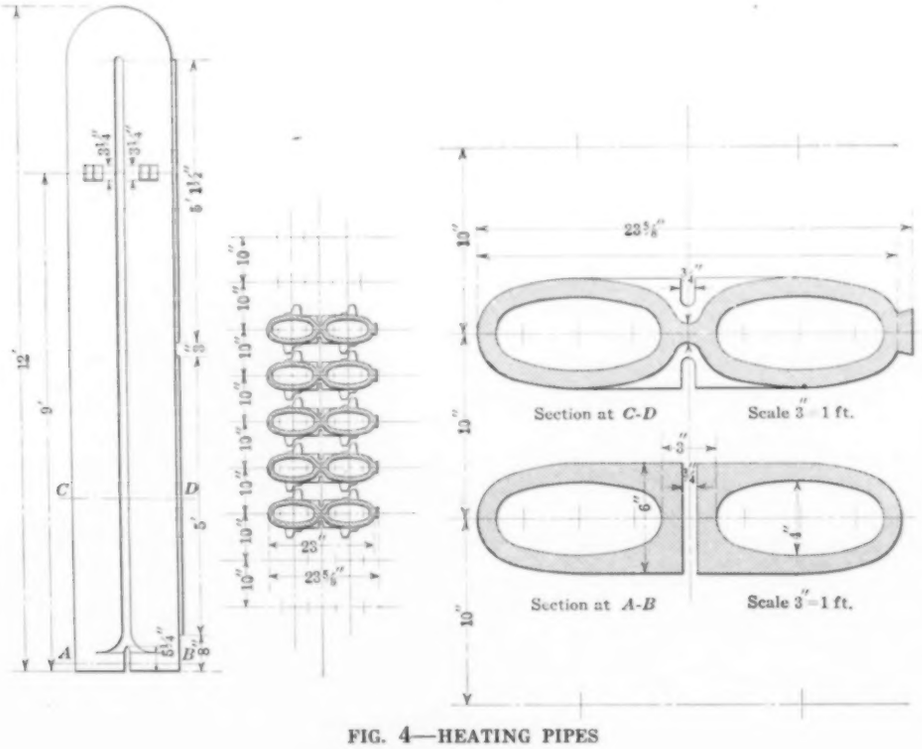


FIG. 4—HEATING PIPES

TWO TYPES OF IRON-PIPE STOVES

Stoves of this general type are divided into two classes, known respectively as the "Glendon" and "Durham," from the former well-known works in the Lehigh Valley, which used and advocated the two types.

There is no essential difference in the construction of these two types, the only difference being in the directions in which the flame and air pass through them. The Glendon stove, as illustrated by the drawings, Figs. 1-4, is on the countercurrent principle. The cold air enters at the draft stack end of the stove and progresses through increasing temperatures to the combustion chamber end, where it is discharged, while in the Durham stove the opposite procedure is followed. This latter at first sight is amazing and contrary to reason, but it appears to be an established fact that higher blast heats can be obtained from stoves of this kind than can be obtained from the Glendon type. The latter

type is definitely limited, if it is to give long life and low repair cost, to 850 deg. Fahr., while from 900 to 1000 deg. may be obtained in the Durham type of stove. This is for the reason that when the counter-current principle is used the pipes next to the combustion chamber carry air already heated up almost to the limit of temperature for the stove, and are exposed on the outside to the flame directly from the combustion chamber. They are, therefore, not cooled off or protected on either side, and if very moderate temperatures be exceeded they will fail in one way or another under this combined assault from within and from without.

In the Durham stove, on the other hand, the coldest air passes immediately into the hottest pipes and cools them down to such a degree that they can withstand the external heat. Of course, in this type of stove the waste gases must pass off at a temperature at least 200 or 300 deg. higher than that of the hot blast they are discharging, otherwise the last rows of pipes would be worthless and would absorb no heat, while in the Glendon type of stove the hot gas may be discharged at a temperature several hundred degrees lower than this and still impart heat to the cold blast just entering.

I do not know of a single case in which any investigation has ever been made of the thermal efficiency of iron-pipe stoves, but I am convinced that that of the Glendon type is considerably higher than that of the Durham. On the other hand, if there be no use for the gas saved by this superior stove economy, as was generally the case at plants where these stoves were in service, then it is much better to burn this gas even though the economy of the operation be low, and convert its heat into increased blast temperature which exercises a beneficial effect upon the fuel economy of the plant as a whole.

Fire-Brick Stoves

The early stoves of this type were of large diameter in comparison with their height, and contained four passes, each passage of the gas vertically through the stove being called a pass. Following the original Whitwell patent a vast number of modifications sprung up and the shape of the stove has undergone an alteration which has now given it a height of four or five times its diameter.

The number of passes has been the subject of experiment, as has almost everything else connected with the stove. It was gradually realized that the fourth pass frequently contributed very little to the heat of the blast, and as every additional pass meant giving up valuable space in the stove for partition walls, with added complexities of brick work at the top and bottom to form safe and durable passageways from one pass to another, the number of passes was reduced to three, to two, and finally to one. Stoves of this type were brought out by Mr. Hugh Kennedy some thirty years ago, which had a single pass straight through from the bottom to short chimneys on the top.

This did away with the necessity of building a chimney, for the stoves had simplified the brick work to the last degree, but the results in practice were unsatisfactory and the stoves did not give the heat desired. They have, therefore, practically passed out of use, and as a result of this and of the gradual disappearance of the four-pass type almost all the stoves built in recent years have been either of the two-pass or the three-pass variety.

The Three-Pass Central Combustion Chamber Stove

The reason for three-pass stoves is that an odd number of passes brings the gas to be discharged at the top of the stove and by adding a relatively short stack immediately on its top we can obtain the draft necessary without the inconvenience of returning the gas back to the ground level and providing a separate stack for it.

The central combustion chamber is for the reason that, as the average temperature of the brick work is higher around the combustion chamber than elsewhere, it is likely to expand to a greater extent there than anywhere else. When the combustion chamber is in the center this excess expansion takes place along the central axis of the stove with complete symmetry in all directions, while the fact that the annular wall which constitutes the outside of the combustion chamber and presumably attains the highest temperature does not support anything at the top, makes it perfectly free to expand and contract without the destruction of any part of the stove. It is claimed in behalf of the central combustion chamber stove that the side combustion chamber locates this unequal expansion unsymmetrically in the stove with a tendency to greater stresses on one side than on the other.

Moreover, the last pass is very much lower in average temperature than the combustion chamber, and in the McClure stove this relatively cool pass extends all around the outside of the stove and tends to cut down the radiation losses below what they are when the combustion chamber is just within the stove wall on one side.

THE M'CLURE STOVE

These two ideas underly the design of the McClure stove, which is one of the best-known types of 3-pass stoves, built by the George W. McClure & Sons Co. This is shown in sectional elevation by Fig. 5, the horizontal section at the level of the hot blast outlet is shown in Fig. 6, a horizontal section of the checker work is shown in Fig. 7. It will be seen from Figs. 5 and 6 that gas and combustion air are introduced at the bottom, pass up through the central combustion chamber, down through the first four rings of checker work to the bottom of the stove, where they pass through arches in its intermediate wall, and so up through the exterior row of checkers to the top, where they pass out through the chimney valve and up the stack. The stove is drawn as "on blast," so the chimney valve is shown shut and the arrows are shown in the reverse direction to that described and illustrate the course of the blast as it passes through the stove and finally out the hot blast opening.

It will be seen that the stove contains two domes, one a spherical one resting on the intermediate wall which separates the second and third passes of the gas and one conical one which lines the conical steel dome of the stove structure and prevents its destruction by heat. Fig. 8 shows a perspective view in section of the lower portion of this stove.

The cold blast valve is a flat bell valve rising against a seat bolted into the top of the stove and renewable through the door in the stack in case of necessity. The cold blast elbow directs the air up against the under side of this valve in order to keep it cool and by this means dispenses with water-cooling for it.

Cleaning doors for the space outside the spherical dome are provided just below the top of the cylindrical portion of the stove, while cleaning doors for the portion inside the stove dome are a short distance below these and give access to doorways through the intermediate wall which are bricked up when the stove is in operation but which may be opened to allow passage into the stove for cleaning.

Clean-out doors are also provided at the base of the stove, opposite the arches which furnish communication between the second and third gas passes.

THE ROBERTS THREE-PASS STOVE

This stove, the design of Messrs. Frank C. Roberts & Co. of Philadelphia, and shown by Figs. 9, 10, 11, 11a and

12, also uses the central combustion chamber principle, but instead of having the second and third passes annular, the space outside the combustion chamber is divided into eight sections by two pairs of parallel walls at right angles to one another whose distance apart is about the same as the diameter of the combustion chamber. This makes four square spaces at the opposite ends of the two principal axes of the stove, and four triangular spaces between these, as shown by Fig. 12.

The intermediate dome of the stove, instead of being a segment of a sphere, is cross-shaped in plan, the tops of the two crosses being arched over with intersecting arches. The gases of combustion pass up through the central combustion chamber, then down through the four square areas within and under this cross-shaped intermediate dome, to the bottom, then up again through the triangular areas, and so to the stack.

The stove, instead of being built of circle bricks and straight bricks laid on the flat, as is the case with the McClure stove, is built up of tiles set on edge and interlocking with one another, as shown by the isometric view, Fig. 12. The checkers are square, instead of having radial ends and circular sides, as in the McClure stove.

This design is used so as to control the gas current to some extent and prevent its traveling too much on one side or another, as it has sometimes been claimed to do in the case of the annular passages of the McClure stove. The cold blast valve and inlet pipe are necessarily located at the top and the current of air is directed against the chimney valve to keep it cooled as in the McClure style.

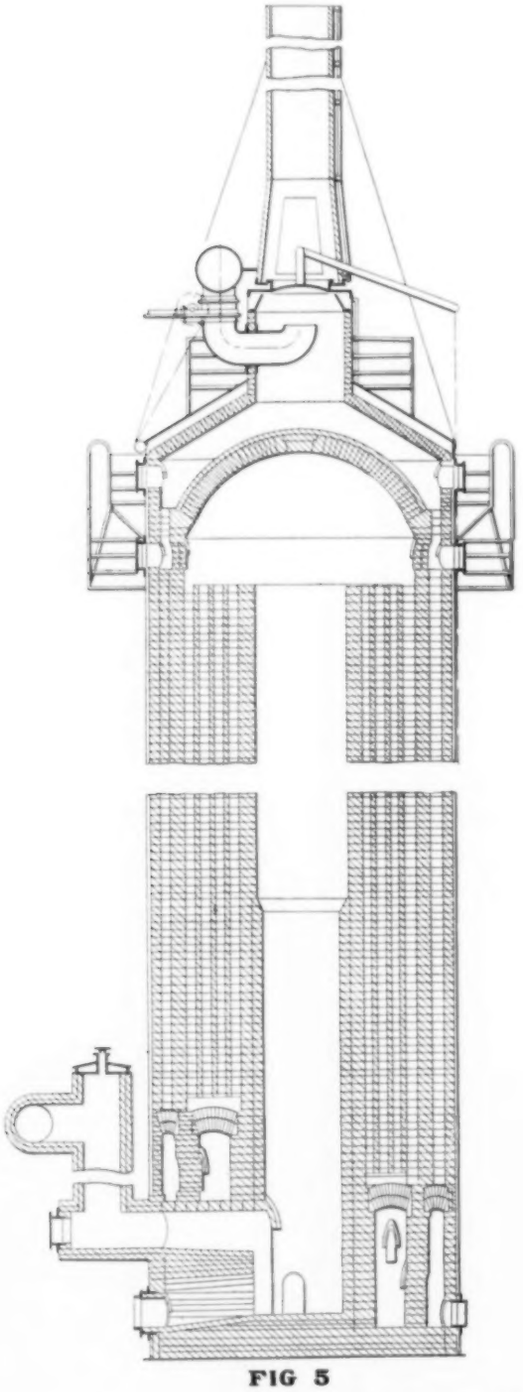


FIG 5

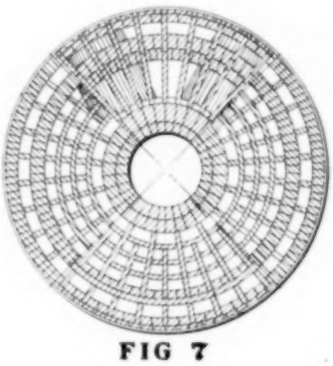


FIG 7

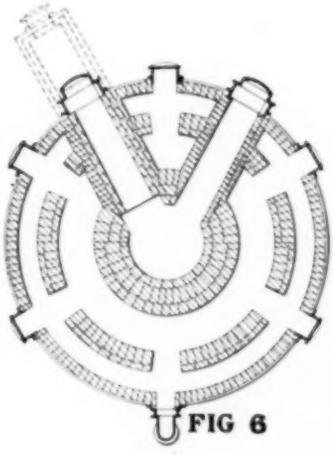


FIG 6

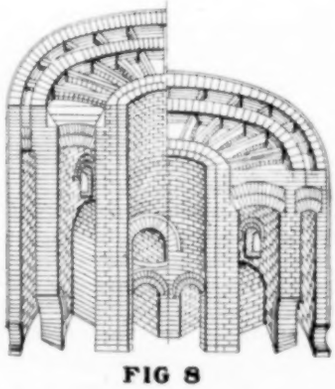


FIG 8

FIGS. 5 TO 8—M'CLURE THREE-PASS STOVE

Two-Pass Stoves

Commonly two-pass stoves have shared the pre-eminence with three-pass stoves in the best modern practice. In the case of the two-pass stove the valves are all at the ground level, so that any blast leakage has at least a fair chance of being detected by the sound it makes, and the interior construction of the stoves is the simplest possible.

Only a single brick dome is required, and therefore the gas in passing from the combustion chamber to the checkers has only to pass over the top of the combustion chamber wall and down. At the bottom the arrangement of flues for gathering burnt gases from the checker work and leading them to the chimney valve, and distributing the blast from the cold blast valve to the checkers is also extremely simple. Moreover, no space is given up to any intermediate wall of any kind.

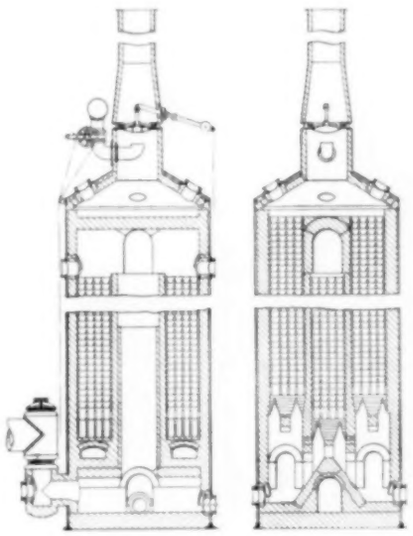
TWO-PASS SIDE COMBUSTION CHAMBER STOVES

Some operators claim that, while the central-combustion chamber is theoretically correct, in practice the side-combustion chamber type gives little or no trouble if properly constructed, and, owing to the fact that the side-combustion chamber is immediately adjacent to the gas and clean-out openings, it is easier to keep clean than is the central combustion chamber because of the distance of the latter from the operator, and the consequent necessity of removing the accumulation of dirt and slag in the center of the stove at a distance of 8 to 12 ft. from the closest point which the operator

can reach. Moreover, the tendency of the blast to wander in the checker work of the two-pass central-combustion type of stove seems to be established beyond question, which is a serious drawback to its efficiency.

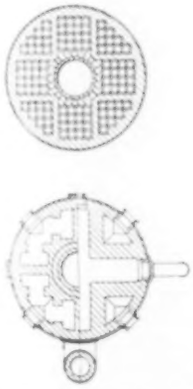
The cross-section of the checkers for a stove of this type of a given size is almost twice as great as that of the checkers of the second and third passes in the McClure stove, and this excess area causes a very low velocity through the checkers, so that a very small difference in draft on one side or another will cause a difference in the flow on that side.

The chimney is necessarily at one side in this type of



FIGS. 9 AND 10—ROBERTS 3-PASS STOVE

stove and the gas passing through the checkers nearest it has the shortest path to the stock, and for that reason the greatest flow will generally take place on that side, with corresponding inactivity in some other region, whose usefulness is thereby greatly reduced. This is for a reason which must never be forgotten in matters of heat conductivity to or from a fluid, particularly a gas, that high velocity of the fluid absorbing or emitting the heat is of the same order of importance as temperature-difference and surface.



FIGS. 11 AND 12 —ROBERTS 3-PASS STOVE

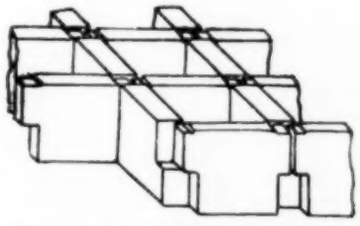


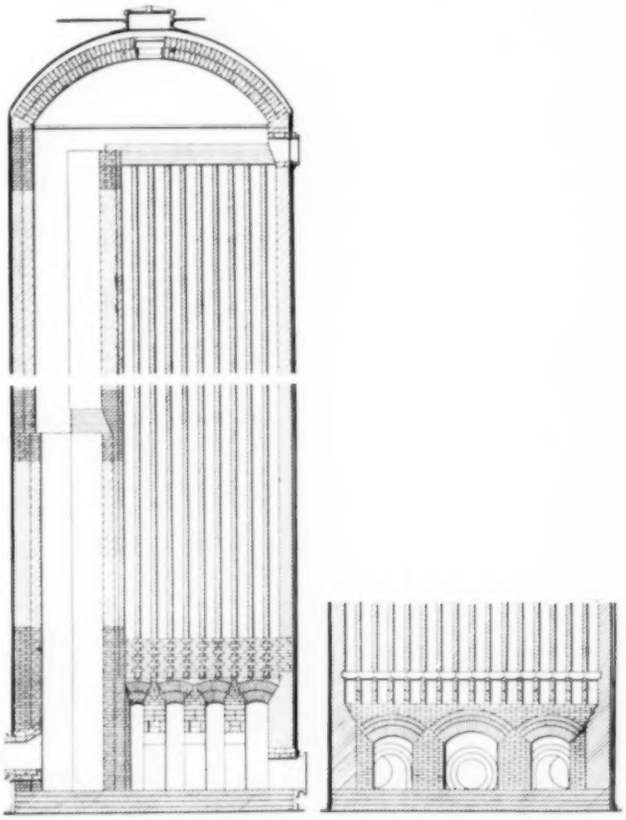
FIG. 11A—ROBERTS 3-PASS STOVE

Efforts have been made to overcome this tendency by equipping each stove with two or even three chimney valves, but while this is undoubtedly good practice, and while two such valves are becoming recognized as correct practice, even for the side combustion stove, it does not, in the experience of some skilled operators, overcome the difficulty completely. Some recent work done in Germany makes it appear possible that we have neglected this question of velocity in the passes far more than its importance warrants and that our stove practice may be materially improved by paying more attention to it.

For these reasons some operators and engineers prefer the two-pass side-combustion chamber stove.

THE NELSON TWO-PASS STOVE

This stove is built by Arthur G. McKee, contracting engineer, Cleveland, Ohio. The principal claims for the stove are that it secures a very large area of heating



FIGS. 13 AND 14—M'KEE NELSON STOVE

surface in proportion to the total cross section of the stove, and that there is a very small percentage of any brick in it, except standard shapes regularly kept in stock by brick manufacturers.

This stove is clearly shown by Figs. 13, 14 and 15. It will be noticed that the back or inside wall of the combustion chamber is given a radius of a curvature the same as that of the front wall. This is a matter of much importance because when this wall has too flat a curvature it becomes a very weak arch and does not resist the thrust due to the expansion of the checker work, and in the past, before experience had taught this costly lesson, many stoves were put out of commission and required to be relined long before they would have been had this point been properly guarded.

The checkers are square in shape and well bonded. and the walls between the flues on which the checkers are built are substantial, so that while these stoves have only been in use for some four or five years, they are well regarded and have given good service.

ROBERTS TWO-PASS SIDE-COMBUSTION-CHAMBER STOVE

In order to meet the views of those who desired a stove of this type Roberts & Co. also furnish one of the design shown by Figs. 16 and 17. This stove uses ex-

actly the same checkers as the Roberts three-pass stove. It is so completely shown by the drawings that no fuller description is necessary.

THE TWO-PASS CENTRAL-COMBUSTION-CHAMBER STOVE

Some operators, while admitting the disadvantage of the three-pass construction and preferring the two-pass

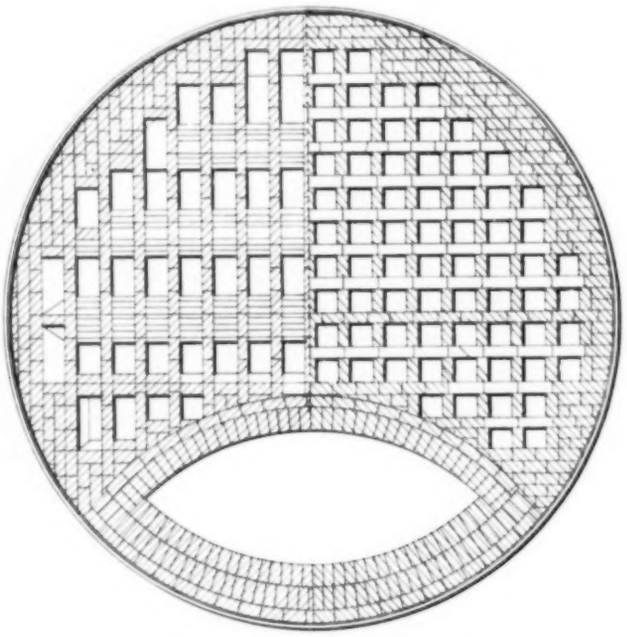
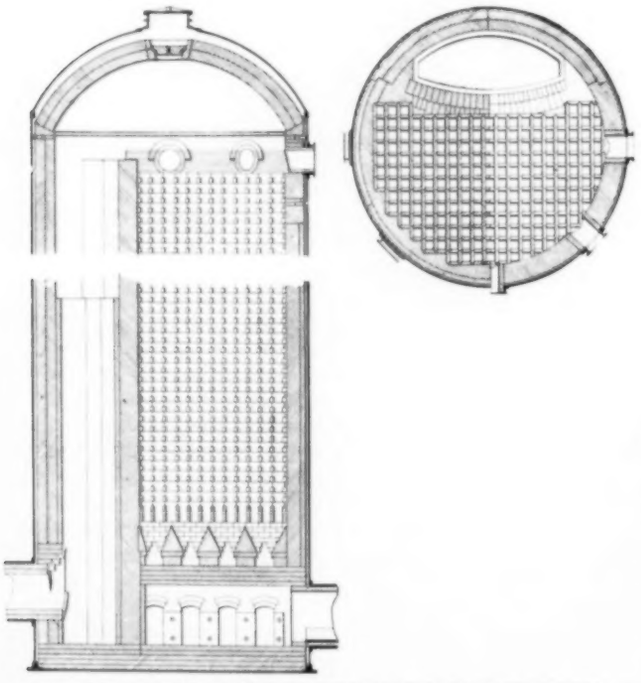


FIG. 15—M'KEE NELSON STOVE

type, consider the central-combustion chamber the best, and a type of stove designed by Frank C. Roberts & Co. to meet this demand is shown by Figs. 18 and 19.

The type of checker work used in these stoves is of



FIGS. 16 AND 17—ROBERTS TWO-PASS SIDE COMBUSTION STOVE

the same type as that used in the three-pass stove, except that, instead of being designed for square checkers, it is designed for radial bricks on two sides and circumferential ones on the other sides. This checker work is shown in perspective view by Fig. 19. The construction of the stove is obvious from the drawings and from what has been said of the principles on which it is based. It is provided with the ordinary accessories in the way of cleaning doors, etc.

Mr. Julian Kennedy was one of the first designers of two-pass central-combustion-chamber stoves and used

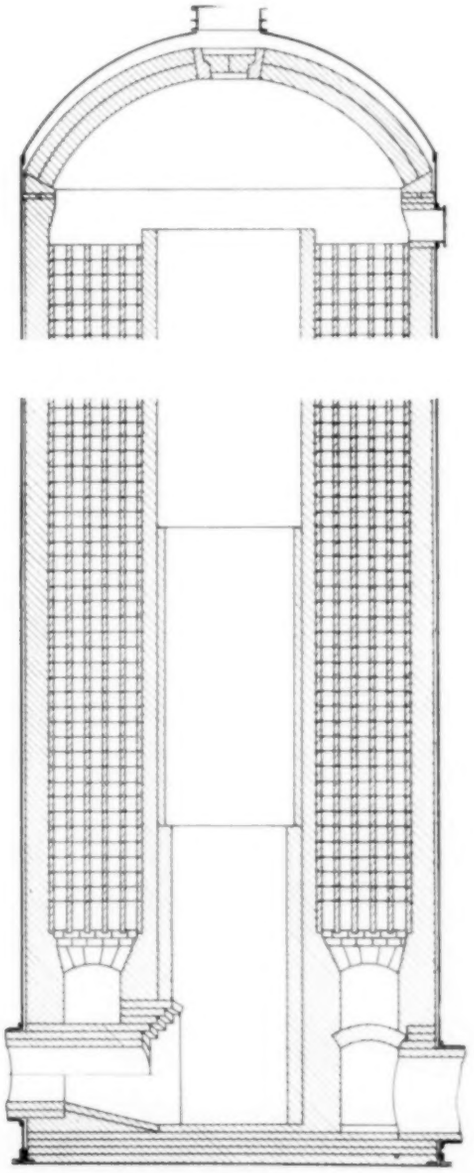


FIG. 18—ROBERTS TWO-PASS CENTER COMBUSTION STOVE

instead of brick checkers hollow key tiles with openings about 9 in. square through the center. The sides of these were radial and the fronts and backs circumferential, so that when stacked up in the stove they made a series of radial checkers of gradually increasing circumferential width passing from the combustion chamber back to the lining of the stove.

The objection to this construction was that a special shape of hollow tile was needed for each ring of checkers and that the partition wall between each two checker openings being made up by the adjacent sides of two of these hollow tiles was split vertically, therefore its strength was not great, and if one of these gave way it was likely to turn over and obstruct the checker completely. If this happened anywhere in the checker except right at the top it was practically impossible to fix it, and that checker was lost to the stove for the bal-

ance of the life of its lining, so such stoves now generally have checkers of brick work built up in the ordinary way.

Stove Values and Fittings

HOT BLAST VALVES

One of the most important details connected with stoves is their hot blast valves. Long experience with

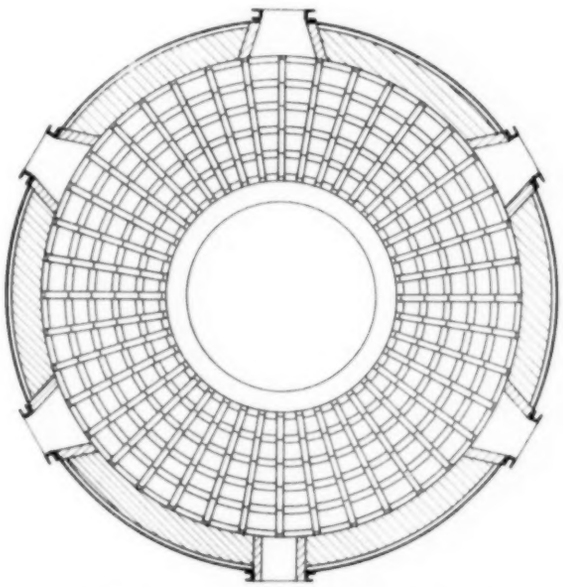


FIG. 19—ROBERTS TWO-PASS CENTER-COMBUSTION STOVE

these has now developed a type which is in almost universal use on all stoves, subject, of course, to individual variation in the details. This consists of a mushroom or a poppet valve resting on a water-cooled seat, the valve being supported by a water-cooled stem. The design used by the McClure Co. is shown by Figs. 22 and 23.

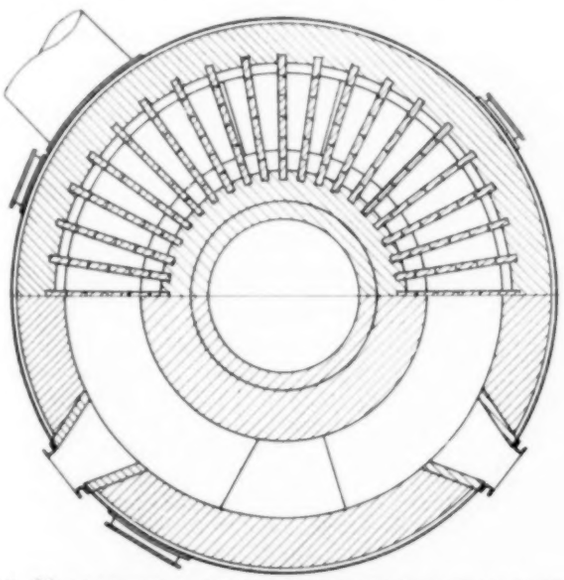


FIG. 20—ROBERTS TWO-PASS CENTER-COMBUSTION STOVE

Commonly the valve is water-cooled as well as the stem, but in the McClure type of hot blast valve this is not done, the valves are made solid without water cooling. The reason for this practice is that if water-cooled valves leak they may discharge a great quantity of water into the hot blast by which it is at once absorbed without showing the least sign, but the effect of this moisture is highly detrimental to the furnace. Moreover, it is claimed by some experienced furnace-men that hot blast valves last about as long without water cooling as they do with it, and that it is better to dispense entirely with the danger of leakage even at the price of a somewhat shorter life of the valves.

The stem, of course, must be water cooled because it must pass through a gland which is required to make a tight joint on it in all positions. This would be quite impossible with the stem heated to redness as it would be if the valve had been for two hours in the closed position. It was formerly the custom to mount the gland immediately on top of the dome-shaped cap of the hot blast valve, but more recently the practice has prevailed of putting in a long sleeve, in the present case about 6 ft. long, which is simply a loose working fit on the valve stem and carries the gland on its top.

The air cooling on the outside of this sleeve prevents the stem from ever getting very warm and the length of the sleeve is such that the portion of the rod which goes inside the hot blast valve proper when it is closed does not come anywhere near the gland when raised into its highest position. This has brought about a great improvement in the conditions of operating and maintaining hot blast valves. It will be seen also that a two-inch air connection is provided at the base of this sleeve whereby additional cooling effect may be secured if desired.

The hot blast valve body is covered with a cast steel cap as shown, and this top is provided with a flange which makes a water box of its upper surface, and the water discharged into this, after cooling the stem, keeps it in good shape. This is, however, protected by an inside cast-steel cover resting on the brick lining of the hot blast pipe. In many cases this cast steel cover is replaced by two large semicircular fire-brick tiles with a hole cut out in their center large enough for the valve stem to pass through, these semicircular tiles resting on the brick work of the lining exactly as shown for the cast cover.

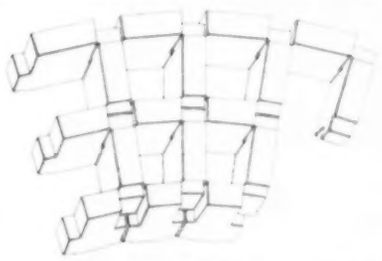
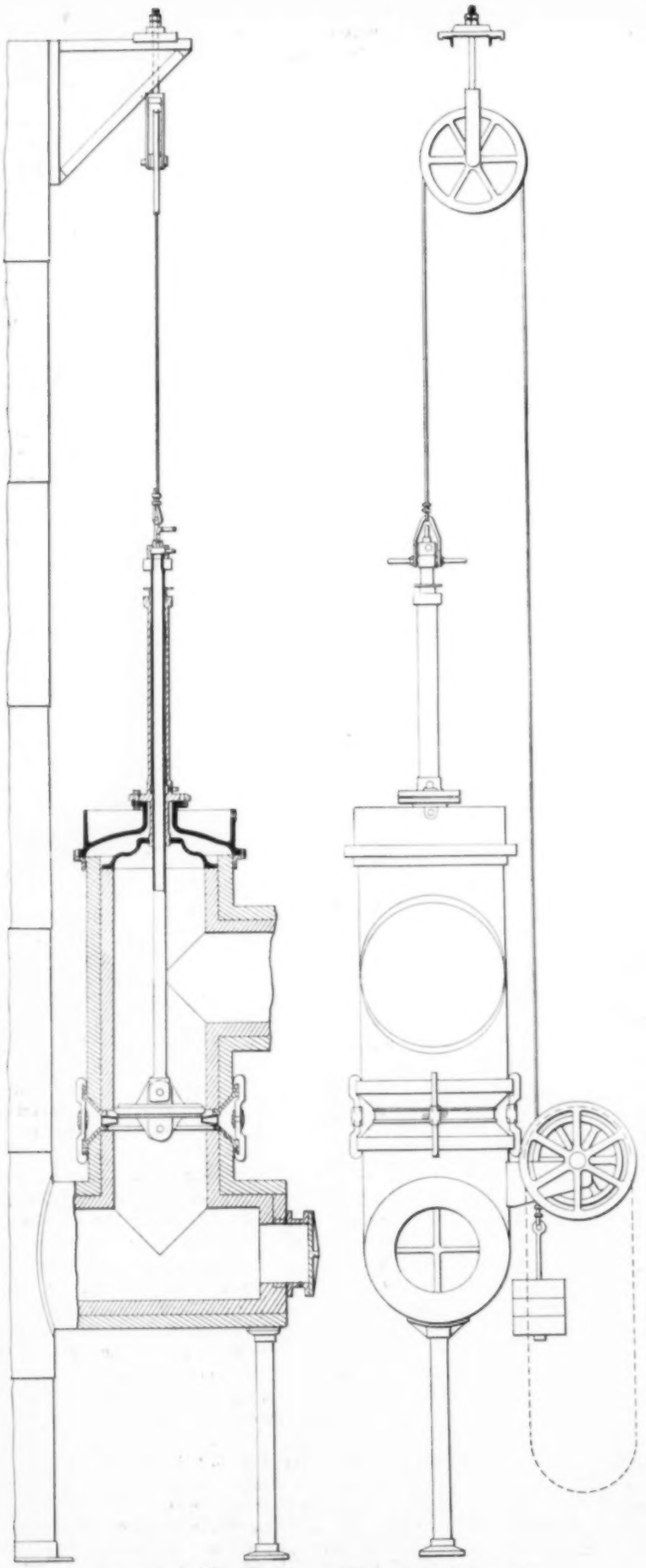


FIG. 21—ISOMETRIC VIEW OF CHECKERS

The water-cooled seat is generally made of bronze, cast hollow to provide for a rapid circulation of water through it. In some cases cast-iron seats with a coil of water pipe cast in them are used, but in good modern practice the bronze seats are becoming almost universal.

It will be seen that the hot-blast valve body is cut entirely in two by this seat and held together by clamp bolts which draw together the top and bottom portions of the valve body so as to make a joint on the top and bottom of the valve seat. This arrangement is used in order to facilitate the changing of these seats, which owing to the high temperature and the abrasive action of flue-dust to which they are exposed are subject to heavy wear and must be renewed quickly, since the blast must be off, and the whole furnace stopped in order to make the change. By slacking off the right and left hand nuts of the clamp bolts and screwing them out until the flat faces on the end of the straight portion of their shanks engage with the flat faces on the seat-rings the latter may be forced apart by an amount slightly greater than the vertical height of the shoulder around the valve seat, and the latter can then be slipped out and a new one quickly inserted. When this is properly centered and the hot blast valve body drawn back to



FIGS. 22 AND 23—M'CLURE HOT BLAST VALVE

place, the projecting ring or shoulder around the seat of the upper flange holds it securely in place.

The hot blast valve is suspended from a rope which passes over a high pulley and back to a drum mounted in bearings bracketed to the side of the hot-blast valve casing. The rope is given two or three wraps around the drum and a counterweight equal to the hot blast valve and stem is attached to the lower end. A hand-wheel on the drum is provided and with this arrangement the stove tender can raise and lower this valve without difficulty. In order to facilitate changing hot blast valves, when this becomes necessary, the high pulley is now, in some cases placed at the top of the stove. This gives ample height to raise the valve, not only to open it, but to lift it clear out of the valve chamber. The high suspension also facilitates swinging the valve to one side in order to lower it to the ground. These are details of much importance when valves have to be changed, since they facilitate doing it quickly. For the same reason the bolts used to bolt the outside cap to the hot blast valve casing should be key bolts, or else the bolt holes should be slotted, so that as soon as the bolt is loosened it may be removed without unscrewing the nut all the way.

THE COLD BLAST VALVE

A standard design for this valve has gradually been evolved, and while the individual valves used by different stove builders differ in details, the general principle of all is the same. They are all flat slide-valves designed to withstand pressure from one direction only, since this is all that is necessary for this service and are always operated by a handwheel and pinion.

Figs. 24 and 25 show the design of the cold blast valve used by Frank C. Roberts & Co. Its construction and operation are obvious from the drawings. There is one point, however, which might escape the casual observer and is therefore worthy of mention. The rack on which the pinion works to operate the valve is not rigidly attached to the latter, but is free to slide between the guides shown in Fig. 25 within the limits set by the stops which come against these guides and through them move the valve proper. The end of the rack is widened out to a T-shape and this covers a slot through the main valve. When the valve is to be opened to put the stove on blast the unbalanced pressure on it is very great, and if the stove tender succeeded in opening it in spite of the friction produced by this pressure, the rush of air into the stove might be so rapid as to do damage. With the arrangement shown the first movement of the pinion slides the rack proper on the back of the main valve and thus uncovers the port in the latter; through this the air flows until in about a minute the pressure within the stove is the same as that in the blast main; this relieves the friction on the valve, which can then be moved back into the valve housing without further difficulty.

STOVE BURNERS

Only one type of stove burner is in extensive use, known from its first inventor as the Spearman burner.

An excellent design of this by the Wm. B. Pollock Co. of Youngstown is shown by Fig. 33. The gooseneck rests on a slide supported on a hollow base which serves as a gas inlet. When the gooseneck is in the position shown, the opening in its base corresponds with that in the slide and its end projects inside the burner opening in the stove so that the gas enters and burns with the air which is drawn in around the gooseneck and also through the small pipe in the center. The purpose of the latter is to give more intimate mixture of air and gas, thus giving better combustion.

When the gooseneck is withdrawn its slide closes the opening in the base and the opening in the stove is closed by a cap or door in exactly the same way as the chimney valve opening is closed in the Stenbakken design. (See Fig. 32 on p. 407.)

Much attention is being paid to stove burners at the present time with a view to securing more perfect combustion by better mixture of the air and gas. Mr. A. J. Boynton, superintendent of the Lorain furnaces of the U. S. Steel Corporation, has put in use an excellent burner which is a decided departure from standard practice. The burner consists of a square casting with a round projecting nose which fits close against the seat ring in the burner opening. This casting is mounted on small wheels so that it can easily be withdrawn to permit closing the burner opening in the regular way. The gas enters at the rear and passes through a number of flat passages cored through the casting. The spaces between these passage ways are about the same size as the passages and open at the sides, these multiple side openings being under the control of a slide on each side so operated as to give the same opening at each air port and to vary the amount from nothing to full opening.

The gas is controlled by an independent bell valve in the supply pipe. This gives complete independent control of both air and gas supply and intimate mixture of air and gas at the nose of the burner.

CHIMNEY VALVES

One of the earliest forms of chimney valves is of the same form as the cold blast valve, but very much larger and water-cooled to withstand the action of the hot chimney gases; its seat is also water-cooled. Such a valve designed by Roberts & Co. is shown by Figs. 26 and 27. Its construction is so clearly shown by the cuts as not to need much description.

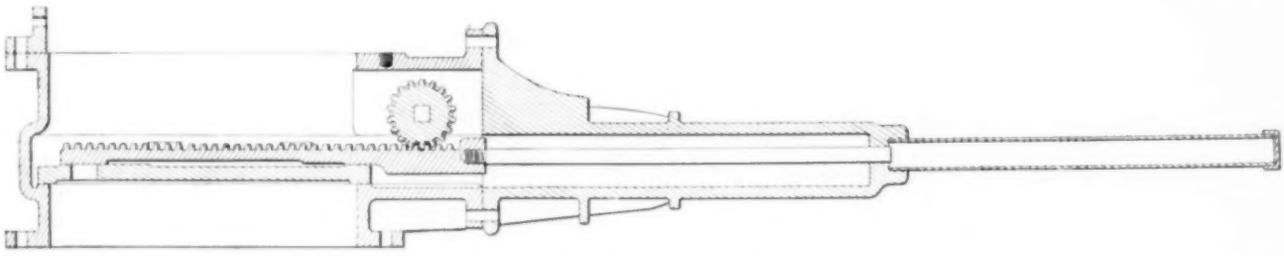


FIG. 24—ROBERTS COLD BLAST VALVE

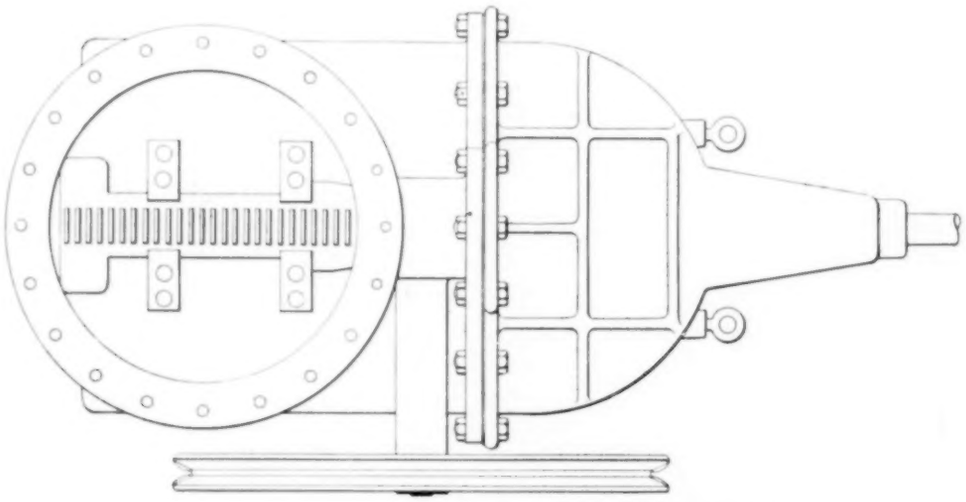


FIG. 25—ROBERTS COLD BLAST VALVE

Like the hot blast valve this valve is counterweighted and operated by a hand-wheel and drum around which the suspension rope passes. The side furthest removed from the stove is turned to a joint on the face and the pressure of the blast when the stove is on blast holds the valve to its seat. When the stove is blown off it then swings comparatively freely and can be raised by the operator without much difficulty.

A mushroom valve of the same type as that used for hot blast has also been largely used in the past. A type of this valve designed by Frank C. Roberts & Co. is shown by Figs. 28 and 29. This also is water-cooled and operated exactly like the hot blast valve.

Valves of these two types, however, have proven very large and unwieldly and rather troublesome in operation, and there has been a strong tendency in the last few years to develop a type of valve which was more accessible, could be more easily operated, and which did not require water-cooling. One of the first and most successful of these was the Kennedy-Morrison valve shown by Fig. 30. This valve is exactly on the principle of the Spearman burner except that it makes as close a joint as possible against the reinforcing ring around the chimney opening in the stove, so as to prevent the ingress of any outside air to the flue. When

the elbow which carries the burnt gas from the stove to the underground flue is withdrawn its slide seals the flue opening exactly as in the Spearman burner, and the opening in the stove is sealed by a cap held to place by one or more screws mounted in swinging yokes, these screws forcing in the ends of a cross-bar or "spider" pivoted to the center of the cap. (See Fig. 31 on p. 407.)

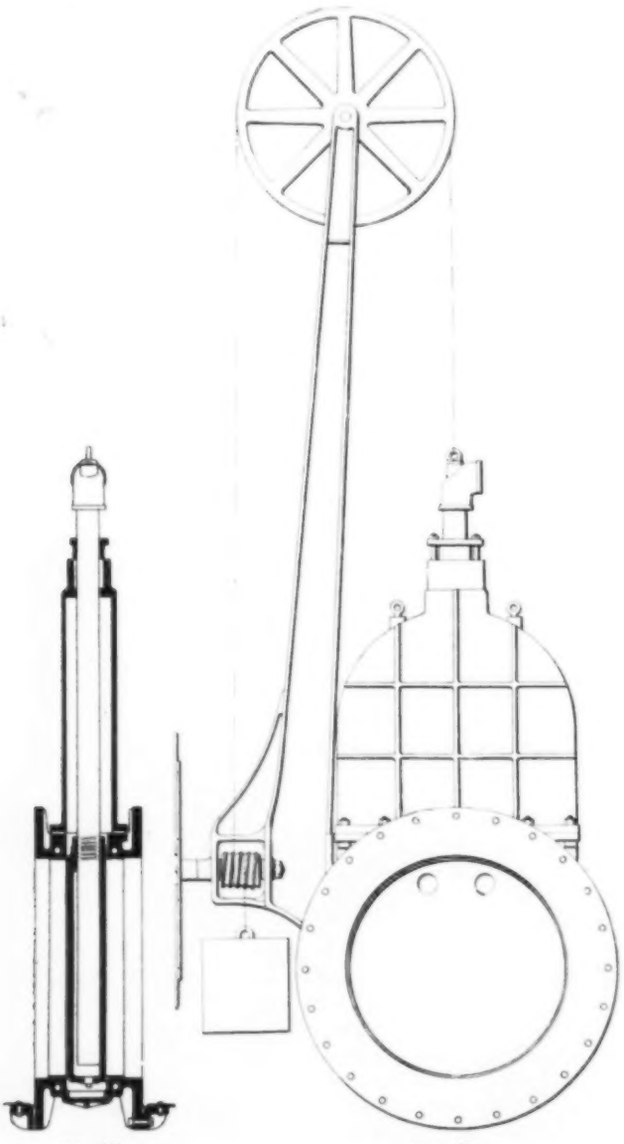


FIG. 27 FIG. 26

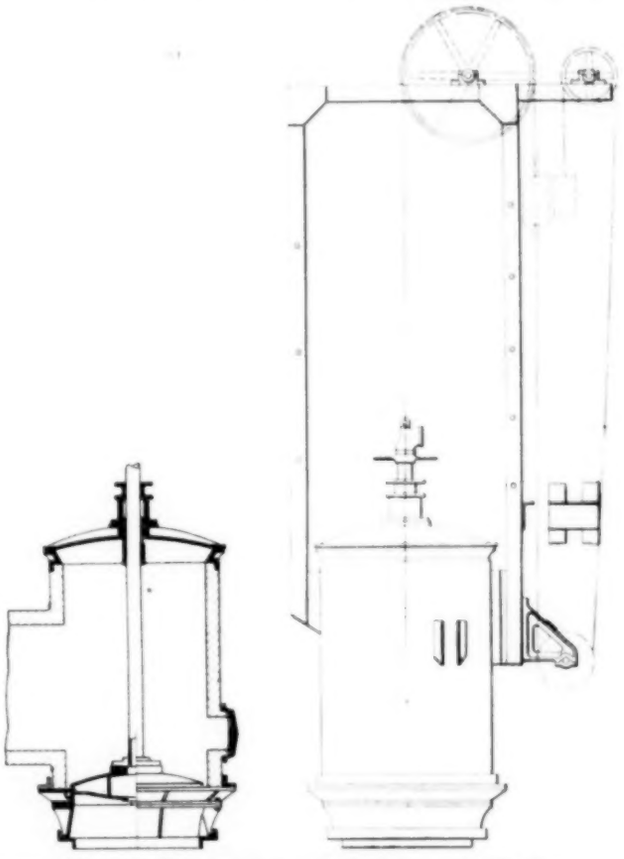
ROBERTS SLIDE CHIMNEY VALVE

The Inland Steel Company have in use a modification of this valve designed by their chief engineer, Mr. Ole Stenbakken, in which the goose neck instead of the sliding back and forth rests on a circular seat, the center of the circle being set off from the central plane of the goose neck proper so that when the goose neck is revolved around this base it is withdrawn from the stove, the opening is then closed by a cap as in the Kennedy-Morrison burner. The construction is shown by Figs. 31 and 32. As these goose necks are very large and heavy it is desirable to reduce the amount of movement and effort required to operate them to a minimum and this is admirably accomplished by the rotation of the goose neck around a vertical axis, its opening, when removed from the stove, is closed by a light steel door.

The chimney valves which are used on the three-pass type of stoves have been shown in conjunction with those stoves and do not need further comment here.

BLOW-OFF VALVES

In addition to the valves above described, there is also an air blow-off valve provided, the function of which is to blow off the air left in the stove when it is taken off the furnace, since, of course, the stove must retain the furnace pressure after its outlet valve is



FIGS. 28 AND 29—ROBERTS MUSHROOM CHIMNEY VALVE

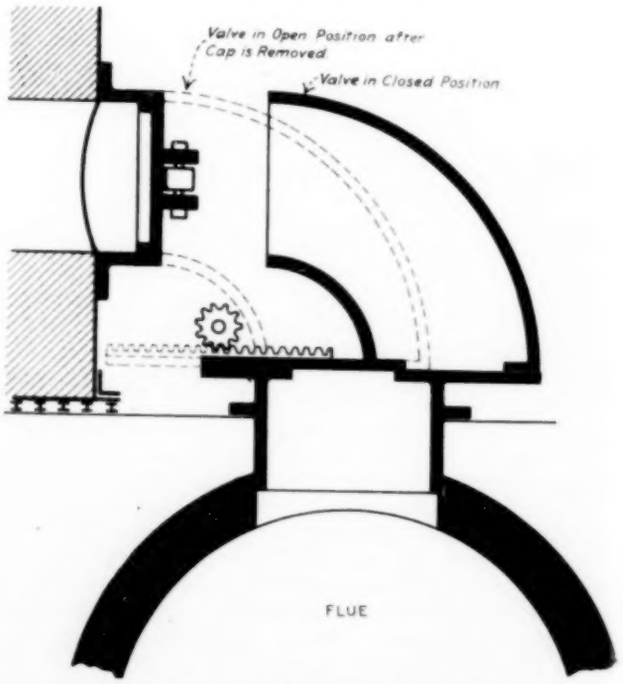


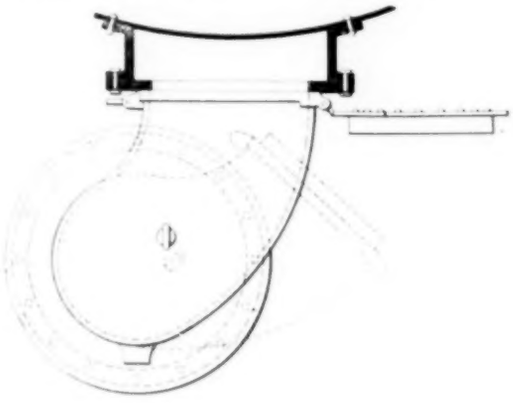
FIG. 30—KENNEDY-MORRISON CHIMNEY VALVE

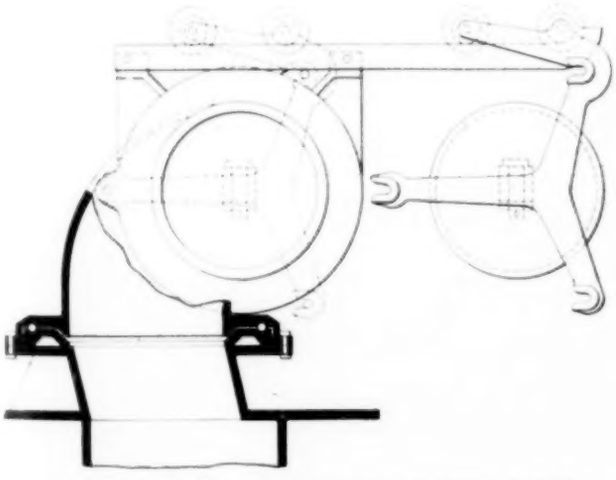
closed. This blow-off valve is placed close to the bottom and is sometimes arranged to open quickly with a lever instead of with a screw, so that as violent a current of air as possible will blow out through it and thereby carry as much dirt as it can from the passages around the base of the stove.

The construction of this valve as used in the McClure stove is shown by Fig. 34. It will be seen that the elbow in which the valve is mounted is arranged to swing as a whole so that this opening serves as a cleaning door as well as a blow-off.

STOVE CASTINGS

In former times the reinforcing rings around doors, valves, burner openings, etc., were made of cast iron, but the liability of cast iron to crack when subjected to expansion and contraction stresses, and the enormous difficulty of getting at the rivets holding these to place





FIGS. 31 AND 32—STENBAKKEN CHIMNEY VALVE

after the stoves are bricked up has brought about the adoption of steel castings for these fittings in all good modern practice. These are somewhat more expensive in first cost, but the failure of a single cast iron fitting after the stove was in operation would involve a loss several times greater than the whole difference in cost between cast iron and steel fittings.

PACKING

It is almost unnecessary to remark that as the steel shell of the stove is exposed to the air and protected by heavy brick work, it remains comparatively cool while the bricks are heated far beyond the red heat. Therefore, the brickwork expands several times more than the shell.

It is also characteristic of brickwork that while it expands when subjected to heat it tends not to contract to the same amount, and therefore is subject to a small but permanent growth. For this reason the shell of a stove must be considerably larger than the outside diameter of the brickwork, and in order to exert a certain amount of pressure against the bricks and keep them to place this space must be filled with some yield-

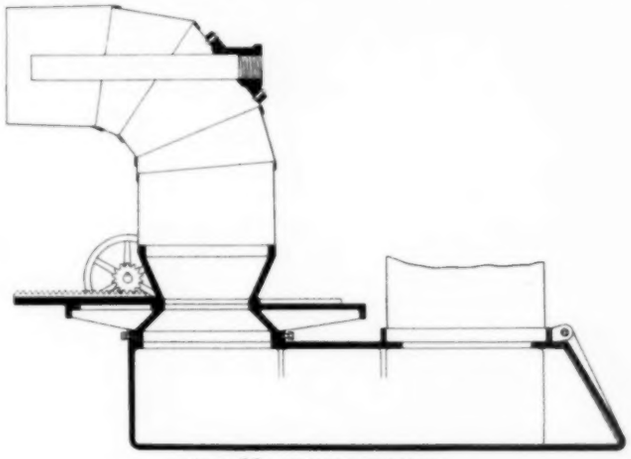


FIG. 33—STOVE BURNER

ing material. Some constructors prefer to use for this purpose pumice-cinder, others clay, and others again a mixture of these materials.

A similar provision must be made for the vertical expansion of the brickwork and the brick dome of the stove must be left many inches below the shell when the stove is lined. There is not the same necessity for packing material in this space as gravity tends to hold the dome to place, and this space may, therefore, be left open without detriment.

STACK FOR STOVES

Mechanical draft has never been applied to hot blast stoves so far as I know, their draft always being supplied by a stack.

The three-pass stoves each have their individual stack

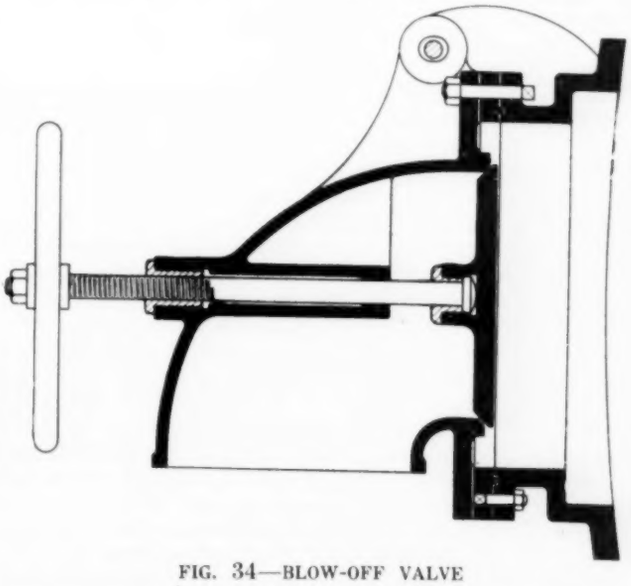


FIG. 34—BLOW-OFF VALVE

resting on top of them. In the case of the two-pass stoves the choice of two styles of construction is possible. One is to have a single stack for a battery of three or more stoves, the entire complement of one or two furnaces. The other alternative is to have a single stack for each stove directly connected with it.

The former practice is very much the more general, and in my judgment much to be preferred, since one large stack is undoubtedly cheaper than three or more small ones of the same total capacity. Moreover, if it be desired to cool off one stove for repairs and it be connected only with its own individual stack, no air can be drawn through it after it has cooled off a little, because there is no difference in temperature to produce a draft, although the stove may be far too hot to work in or even to enter. But if all the stoves are connected to a common stack then the chimney valve on the stove which it is desired to cool off may be opened to some extent without seriously impairing the draft of the other stoves and the cold air thus drawn through it will cool it down to a temperature at which it can be entered and repaired in a small fraction of the time in which it would cool to the same extent without this assistance.

THE DOMES OF STOVES

In the early days many stoves were fitted with a cylindrical top as shown by Fig. 35, and this shape

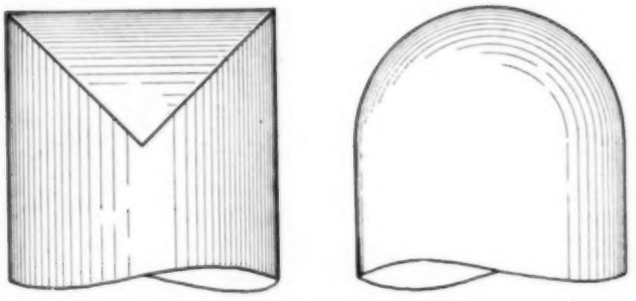


FIG. 35—CYLINDRICAL STOVE TOP

conforms with the mechanical requirement that all sections of the stove shall be circular in order to resist deformation, but this style of top is difficult and expensive to fit and seems particularly liable to give out along the square corners where the horizontal cylindrical surface of the top meets the vertical surface of the stove body, and stoves are now almost universally fitted with either conical or spherical tops.

THE BOTTOM AND BASE RING OF STOVES

The bottom sheet of the stove must of necessity carry all the brickwork in the stove and is, therefore, not capable of repair except by tearing out practically the whole lining of the stove. The principal point of corrosion about stoves appears to be where the cylindrical surface meets the bottom plate, and owing to this corrosion old stoves are liable to fail at this point sooner than almost any other, the base ring contains nearly all the openings made into the stove for cleaning out doors, valves, etc., and its strength is thereby greatly reduced. For these reasons the bottom section should be made of heavier material than the rest of the stove.

The stove base instead of being set down flat on a masonry foundation, as formerly, now generally rests on a set of T-rails placed close together, this carries it up off the ground and prevents corrosion to a large extent.

The joint where the bottom sheet meets the side sheets should always be made with the utmost care and almost without regard to expense. It is very doubtful if flanging is a good construction at this point. A heavy angle bent to the circle and riveted to the bottom and side sheets furnishes a better construction.

THE CHECKERS IN STOVES

It has been the practice to make checkers in stoves about nine inches square in the past, but this practice is now beginning to undergo a change which is likely to be very rapid, for the reason that until quite recently gas cleaning was not practised to any considerable extent in this country, while the blast furnace gas produced here probably carries more dirt and fine ore than that of any other country, because of the predominant portion of fine sandy ore used.

This dust is carried into the stove by the gas and deposits on the brickwork which, of course, is composed largely of silica; under the intense heat to which this brickwork is exposed the iron ore constituting the larger portion of the flue dirt is fused to the brickwork and the wall of the combustion chamber soon becomes glazed over and finally is slagged away by this action, so much so that a false wall is always built in the combustion chamber, so that this wall may be renewed when it is burnt out without affecting the stove as a whole. In one plant, formerly under my management, we never counted that these walls would last over eight or nine months without repair, and this was due almost entirely to the slagging action of the iron ore carried by the gas.

In the checkers the conditions are not quite so bad, because the temperatures are much lower, but vast quantities of dirt settle on these in light flocculent masses, about the best non-conductor of heat which could be found for such high temperatures. These flocculent masses greatly reduce the flow of gas through the stove and seriously impede the passage of blast through it also. Therefore, it is now necessary at intervals to take each stove off the furnace for several days, cool it down so that it can be entered, put men in the upper dome and clean these checkers by dropping through them weights suspended from a rope or chain. This is a highly laborious performance, and from the nature of the case more or less ineffective. In order to carry out this operation at all, it is necessary to lose the use of the stove for from one to two weeks under the best conditions, and during that time, unless the furnace is more than usually well provided with stove capacity, the blast is improperly heated, and the fuel economy of the furnace suffers.

In recent years it has been found that with the exercise of a certain amount of care the gas can be scrubbed practically free of these impurities at comparatively small expense. This eliminates the necessity of allowing for a coating of flue dust from one to three inches thick on all sides of the checkers. When this is done it becomes a simple matter to use checkers six inches square which are not only as good as, but much better than checkers nine inches square when dirty gas is used.

It is obvious that the amount of heating surface which can be put into a given stove with six-inch checkers is far greater than that which can be put into it with nine-inch checkers and this means that the stove capacity needed to supply a given amount of heat to the blast may be more cheaply obtained, not only in operating cost but also in first cost by putting in a gas-washing plant to supply clean gas, and using somewhat smaller stoves with smaller checkers. The smaller stoves so designed contain more heating surface and maintain a higher efficiency than the larger stoves under the old conditions with dirty gas.

The Number of Stoves Per Furnace

There is no theoretical reason why one iron-pipe stove should not be sufficient for a furnace, because it is an apparatus which works continuously by the conduction of heat through metal walls exactly as a boiler does. Barring repairs it could run on indefinitely, and small furnaces in the early days were frequently run with only one stove.

As a matter of fact, especially at large furnaces, the number of stoves is increased for two reasons: First, it is difficult to build a single unit large enough to heat the quantity of blast required for a large or even a moderate-size furnace, and at the same time to design it so as to give good control of the heat in all parts. Second, after a certain point is passed it costs no more to build the required stove capacity in two or three units than it does in one, while if there be, say, three stoves on a furnace, then, when one of them requires repairs the other two are still able to heat the blast to a very valuable extent; whereas, if only one stove be provided, when this goes down for repair the furnace must shut down because it is commercially impossible to run a furnace on cold blast.

For these reasons two or three iron-pipe stoves are commonly provided per furnace when that type is used.

In regard to the capacity and efficiency of these, little or nothing is known. As a matter of experience it can be stated that the stove illustrated in Figs. 1, 4 is capable of heating six or eight thousand feet of blast per minute up to 850° F. without such overcrowding as will destroy it too rapidly. In regard to economy and efficiency we have not even the guide of general experience to help us, because there is no way to get at the quantity of gas consumed by one of these stoves without careful experiments on gas flow, etc., which, as far as I know, have never been conducted.

It may be safely stated that the efficiency of the Durham type of stove, that is the parallel-current type, is quite low, because the chimney gases must be discharged at a temperature of twelve hundred or more, and even with theoretically perfect combustion this involves a loss of over 50 per cent. of the heating value of the gas.

Turning now to the fire-brick stove, the conditions are entirely different from what they are in the case of the iron-pipe stove, since it is impossible to get along with less than two of these, one of which is being heated up while the other is giving up its heat to the blast. Furnaces were originally operated with only two stoves of this type, but this condition has long since passed away. Less than three are never contemplated, four is only ordinary good practice, and five have been installed with extremely satisfactory results at some plants.

The reason for the increased number of stoves is twofold. First, to have one in reserve, or at least be in a position to spare one for repairs without crippling the plant; second, because the heating of the blast by this process is an intermittent operation, and therefore the temperature of the blast fluctuates with the condition of the stove. When the stove is first put on it has been heated up to a high temperature, which falls as it gives up its heat and the blast temperature is therefore constantly falling.

When only two stoves are provided these variations are so great as to cause serious irregularities in the work of the furnace. By adding a third stove and burning the gas so as to give them what steel men know as a "soaking heat," their temperature falls much less rapidly while they are on blast. The fact that the relative time on gas is doubled while the time on blast remains the same contributes much to this result. This greatly reduces the irregularities due to the intermittent action of the apparatus.

But, when one stove requires to be taken out of service for repair the furnace is then thrown back onto the two-stove condition. This cuts down the heating surface to two-thirds and reduces the average blast temperature materially as well as making serious irregularities in the temperature curve. Therefore, in order always to have the benefits of three stoves four must be provided; this gives three hours on gas for one on blast, which is a step better than the condition existing when three stoves are used.

Four stoves introduce the possibility of operating along a somewhat different plan, with the object of eliminating inequalities of blast temperature still more. This plan is to blow through two stoves simultaneously, leaving them each on blast one hour, but changing them at alternate half hours. This reduces the time of the stove on gas to one-half the total time, but halves the amount of heat taken out of it at any period. It involves somewhat more work for the stove tender, but practically halves the variations of blast temperature from maximum to minimum.

Here, again, we come to the necessity of occasional repairs necessitating a return to three-stove conditions, and this has caused many operators to consider five stoves a good investment, because then four stoves can always be maintained in operation, even when one is being repaired, and when all five are available each one can be left on gas for three hours and on blast for two hours, with still better results in eliminating irregularities in the blast temperature and maintaining a high average temperature without driving the stove too hard when it is on gas.

It will be seen, then, that the number of firebrick stoves to be supplied per furnace depends to some extent upon natural conditions, which we cannot control, and that if the best results are to be attained under present conditions, not less than four must be supplied. But, since the introduction of clean gas at some plants, it has been found that the stoves do not need to come off during an entire blast for either cleaning or repair, and this fact in addition to the increased economy and greater heating surface attainable as a result of clean gas have a tendency to reduce the number of stoves required, and it may well be doubted if five stoves will in future be installed at plants originally designed to use clean gas. It may be that the number required will fall to three.

EQUALIZING THE BLAST TEMPERATURE

The proposal has been made by several blast-furnace engineers and operators to apply an equalizer to remove the objectionable peaks from the charts of blast temperature, and several of these apparatus have been built. One of the few in this country was that built by John S. Kennedy and the late John M. Hartmann at the Musconetcong furnace, Stanhope, N. J., which has been most successful in operation.

This apparatus is practically a small stove without combustion chamber or valves filled with very small checker work through which the blast passes on its way from the stoves to the furnace. When the blast is very hot from putting on a fresh stove the excess heat above the average is given up to the checker work in which it remains stored until the temperature of the blast falls below the average, owing to the cooling of the stove, when this stored surplus flows out and makes up the deficiency.

Mr. Kennedy has told me that this apparatus smoothed down the "saw teeth" on the blast temperature charts so that the maximum temperature difference for long periods continuously was less than twenty-five degrees. Without such an apparatus the drop in temperature between putting on and taking off a stove will run from 50 to 150°, depending on the number of stoves and how they are handled.

Another method for obtaining a uniform blast temperature is by the use of a cold-wind mixing valve. This valve is located on the hot blast main into which it opens and is connected with a branch from the cold blast main. By opening this valve cold blast directly from the engine can be admitted into the hot blast

main, thus cutting down the temperature of the blast by admitting a considerable quantity of this cold air as a fresh stove is put on the furnace and gradually diminishing this quantity down to nothing, at or near the time when the stove comes off.

By this means the blast temperature may be maintained uniform, but this method is open to two objections: First, that the uniform blast temperature is, in that case, the lowest temperature furnished by the stoves instead of the average, and, second, the equalization depends wholly on the skill and fidelity of the stove tender, and if done properly requires such frequent attention by him as militates against his assisting at other work.

Attempts have been made to provide an automatic control of this valve, which would relieve the stove tender of the duty of regulating it and would give perfect equalization of the blast temperature, but so far as known to me none of the apparatus designed for this purpose are now in regular commercial use.

The Size and Heating Surface of Stoves

Practice in regard to these matters in the past has been based on general experience and the belief of the furnace man or engineer that stoves of about such a size would be sufficient to supply the hot blast for a given furnace. It is not a rule, but it seems to be according to general practice that the diameter of the stove shall be about the same as the bosh diameter of the furnace, and that its height shall be as a minimum that of the furnace and from that up to twenty or twenty-five per cent. more.

Until recently but little attention was paid to the quantity of heating surface furnished per foot of stove height, but there is evidence that this condition is passing and that within a few years stoves will be bought on their heating surface, practically as boilers are today. This has been impossible except in the most general way in American practice until now, because the hard driving of American furnaces, combined with the enormous percentage of extremely fine ore used at seventy-five per cent of them, has made an exceedingly dirty gas, and this dirt depositing on the checkers in the way already described has reduced the efficiency of the stove in proportion to the extent in which it has been allowed to accumulate, so that stoves kept clean and under good management at one plant are capable of heating a given quantity of blast to a much higher temperature than similar stoves at another plant where they are not properly cleaned.

To a large extent these conditions have been past control and absolutely past all calculation. General experience has, therefore, been practically the only guide. With the introduction of primary cleaning for all the gas, that which goes to the stoves will be in vastly better condition than it is now, the stoves will stay clean for far longer periods and will have not only a higher but a more definite efficiency.

The advantages of smaller checkers already pointed out, which become available under those conditions, will cause competition among stove builders to give the maximum amount of heating surface per stove, and we shall gradually reach the place where the capacity and economy of stoves are as well understood as that of any other apparatus around the place, a condition which certainly does not exist at the present time.

There has been in recent English and American literature little or nothing on the hot blast stove from the operating point of view, but in December, 1912, Mr. A. N. Diehl published before the Engineers' Society of Western Pennsylvania a very valuable paper giving the results of long and careful investigation of the subject, and in October, 1913, in a discussion of the sub-

TABLE NO. 1.
EXCERPT FROM STOVE TESTS.

Stove.	Heating surface sq. ft.	Volume of brickwork cu. ft.	Fuel gas.	Volume of fuel gas per Min.	Calorific value of fuel gas B.T.U.	Blast per Min. cu. ft.	Temperature flue gas.			Temperature blast.			Heat balance.		Stove on gas. Mins.	Stove on blast Mins.	Flue gas percent	
							Max.	Min.	Avg.	Max.	Min.	Avg.	Absorbed blast. Percent	Lost in flue gas Percent			CO	O_2
1 Center Comb. 3-Pass	49865	20831	Clean	4252	108.5	36228	680	370	630	1565	1270	1418	63.37	17.53	172	58	0.47	2.29
2				4527	105.8	35742	720	565	670	1600	1360	1462	62.54	17.40	179	60	0.12	1.13
3				3447	99.0	36542	650	540	619	1440	1235	1328	61.59	17.65	180	60	2.40	
4				3396	104.4	36109	640	540	614	1335	1136	1231	66.02	16.87	181	59	0.11	2.94
Average.				4010	104.4	36130	685	554	633	1487	1250	1359	63.30	17.84	178	59	0.77	1.65
1 Center Comb. 2-Pass	39220	17974	Clean	3838	104.3	40030	475	370	442	1310	800	967	61.75	24.64	175	58	0.60	8.00
2				3523	94.1	33905	670	495	625	1440	1085	1200	53.28	28.70	171	52	0.60	6.50
3				4497	102.1	37678	700	490	624	1500	850	1051	44.03	34.17	179	50	0.80	9.70
4				3548	104.6	38599	650	500	624	1440	1036	1156	61.24	20.14	172	61	0.90	4.60
5				3489	94.3	40530	525	420	441	1351	1076	1201	64.48	19.28	199	64	0.80	2.90
6				2805	99.7	34343	680	485	612	1411	1020	1176	66.73	13.16	181	60	1.10	2.60
7				3156	96.6	39632	535	430	489	1340	1010	1168	65.26	12.33	190	57	2.00	
8				3092	99.2	38852	525	420	476	1470	1030	1149	61.98		182	57	3.30	
Average.				3368	99.3	37951	595	451	540	1358	969	1094	58.60	20.40	181	57	1.26	4.27
1 Center Comb. 2-Pass	39220	17974	Dirty	4120	103.2	39149	860	610	788	1460	1015	1163	47.13	29.50	200	66	0.80	5.60
2				4735	94.9	39000	915	650	819	1460	1000	1203	54.20	24.76	190	66	0.80	6.60
3				4241	105.0	38735	960	620	864	1410	990	1179	50.68	24.20	169	51	0.80	1.00
4				3898	99.4	38780	930	600	813	1425	1000	1148	56.97	24.95	149	51	0.60	
Average.				4248	100.6	38916	916	620	921	1336	1001	1176	54.24	25.85	177	59	0.78	1.80
1 Side Comb. 2-Pass	30725	7494	Dirty	2832	79.6	35802	310	340	457			1047	42.30		179	30	0.82	4.47
2				3114	84.1	35802	550	295	476			982	59.3		174	53	0.24	1.56
3	21854	5330		3061	82.6	35602	430	320	377			944	57.90		157	45		
4				2448	82.1	35802	730	305	467			974	64.90		196	49	0.26	2.54
5				2772	75.8	35802	740	390	478			891	62.30		173	51		
6	33632	8204		3524	83.2	36400	650	495	600			986	56.10		180	54	0.17	3.58
7				3857	87.9	36400	690	450	606			990	58.20		165	60		1.77
Average.				3091	82.7	35990	583	370	509			977	57.30		175	49	0.37	1.99

Note: Side Combustion Stove as above, volume of brickwork in fire checkers only.
Side Combustion Stove as above, volume of brickwork in fire checkers and combustion chamber.

TABLE NO. 2.
SHOWING HEAT LOSSES IN STOVE PRACTICE DUE TO INEFFICIENT COMBUSTION OF FUEL GAS.

Test No.	Volume of fuel gas per Min. cu. ft. 1	Volumetric ratio flue gas to fuel gas. 2	Volume of flue gas per Min. cu. ft. 3	Calorific value of fuel gas B. T. U. 4	Average temp. flue gas. 5	Flue gas. Percent. CO 6	Flue gas. Percent. O_2 7	Heat losses in flue gas due to CO 8	Heat losses in flue gas due to O_2 9	Volume of CO in flue gas per Min. cu. ft. 10	Volume of excess air in flue gas. cu. ft. 11*	Total heat losses due to CO & excess air. Percent 12	Coke eqv. of heat losses, per hour. Lb. 13†	Coke eqv. of heat losses per day per Pce. Lb. 14‡
1	4282	1.993	8534	108.5	630	0.47	2.29	2.46	2.15	30.11	862.51	4.61	106.3	7582
2	4527	1.718	7777	105.8	670	0.12	1.18	0.53	1.07	9.23	416.31	1.60	37.7	2714
3	3847	1.640	5924	99.0	619	2.40		11.61		142.18		11.61	217.5	15660
4	3386	1.849	6260	104.4	614	0.11	2.94	0.50	2.70	6.89	849.91	3.20	56.5	3996
Average	4010	1.775	7049	104.4	633	0.77	1.46	3.77	1.97	47.12	709.57	5.24	104.0	7763
1	3838	2.925	11235	104.3	442	0.60	8.00	4.70	8.21	67.43	4136.25	13.01	256.1	18439
2	3523	2.000	7363	94.1	625	0.60	6.50	5.15	7.53	44.18	2182.09	12.68	207.3	14925
3	3497	2.574	9001	102.1	615	0.80	9.70	8.21	12.72	72.06	3908.68	21.03	269.1	26575
4	3548	2.078	7373	104.5	624	0.90	4.60	5.37	4.54	66.26	1461.72	9.91	221.7	15962
5	3489	1.757	6130	94.3	612	0.80	2.90	4.52	2.60	36.78	761.72	7.12	115.2	8294
6	3505	1.805	5063	99.7	441	1.10	2.50	6.32	1.24	55.69	471.92	7.36	110.2	7934
7	3156	1.291	4043	96.6	499	2.00		9.39		80.66		9.39	140.8	10137
8	3092	1.558	4817	99.2	476	3.30		14.34		158.96		14.34	216.6	15595
Average	3365	2.134	6853	99.3	540	1.26	4.27	7.12	4.62	71.53	1626.54	11.87	204.6	14733
1	4120	2.415	9950	103.2	788	0.80	5.60	5.32	8.43	79.60	2473.38	13.80	288.5	20772
2	4735	1.884	8731	94.9	819	0.80	0.60	4.11	0.28	69.85	83.74	4.39	97.0	6984
3	4241	1.731	7341	105.0	884	0.80	1.00	3.75	0.80	58.73	210.62	4.55	100.0	7200
4	3898	1.654	6447	99.4	813	0.60		2.87		32.68		2.85	54.2	3892
Average	4248	1.911	7867	100.6	821	0.75	1.80	4.01	2.39	60.21	691.98	6.40	134.9	9712
1	2832	1.834	5194	79.6	457	0.82	4.47	6.23	3.71	42.59	1008.16	9.94	110.4	7935
2	3114	1.546	4814	84.	475	0.34	1.56	1.45	1.09	11.55	330.72	2.54	32.8	2362
3	3261			82.6	377									
4	2448	1.718	4205	83.1	569	0.26	2.54	2 08	2.92	10.93	454.42	5.00	43.6	3067
5	2772			75.8	476									
6	3524	1.778	6266	83.2	696		3.38		3.1:		1072.12	3.16	58.0	4176
7	3887	1.574	6118	87.9	606	0.17	1.77	1.01	1.60	10.40	491.28	2.61	43.7	3146
Average	3091	1.690	5319	82.7	549	0.21	1.99	2.15	2.49	15.07	757.44	4.65	57.5	4137

*Col. No. 11—Over and above that required to oxidize the CO present in the flue gas.
†Col. No. 13—Per stove per hour and based on coke having a calorific value of 13 200 B. T. U. per lb. as received.
‡Col. No. 14—Based on four stoves per furnace, or 72 stove hours per day on gas and 24 stove hours per day on blast.
Note: All gas volumes and calorific values are at Standard Conditions, 62 F. and 30 in. Hg.
Above test Numbers correspond to those given in Table No. 1.

ject of gas cleaning before the American Institute of Mining Engineers he contributed additional data on the subject.

His tables I. and II. from the first-mentioned paper are reproduced here and will repay careful study.

With clean gas the efficiency of the stove is raised considerably, according to Mr. Diehl's later figures, from 56 per cent. to 64 per cent. This is not a decrease of 8 per cent, but of $12\frac{1}{2}$ per cent (8 ÷ 64) in the gas consumption of the stove and, what is much more important, it brings with it the possibility of obtaining higher blast temperatures with the given equipment of stoves and by this means reducing the coke consumption of the furnace.

Table I. shows the advantage of having a sufficient amount of heating surface for the given conditions and as the use of clean gas makes permissible the use of smaller checkers and thereby increases the heating surface, it may be said that the effect of clean gas is cumulative in its influence on stove design and operation. The best practice as brought out by Mr. Diehl's paper and its discussion favors the use of not less than four and preferably five square feet of heating surface per cubic foot of blast blown per minute measured by piston displacement. When four stoves are used this would mean from one to one and a quarter square feet of heating surface per stove for each cubic foot of blast blown per minute.

This will obviously vary to some extent with the conditions. When high heats are regularly required, as with the use of lean and refractory ores, more heating surface should be provided than with soft and reducible ores which ordinarily do not need such high heats. On the other hand, when there is no use for the surplus gas it may be better to save in the size of the stoves and burn the gas less economically in them to get the desired heat.

Since this article was written there have appeared accounts of recent experiments in Germany in which the heat imparted to the same was greatly increased by increasing the velocity of the gas through it, first by an air jet in the combustion chamber and in the later experiment by use of a fan to increase the draft. It is claimed that by the use of this system so much heat can be stored in each stove that only two are needed per furnace.

The accounts are too incomplete, and it is too early in the development to say what the final result will be, but with the vast improvement made in the operating conditions of stoves by clean gas there is no doubt that many improvements will follow and that increasing the velocity of the gas and blast through the checkers is one of the best ways to increase the rate of heat transmission.

As the efficiency of stoves increases it is very likely that fans will be used to provide draft for them for the reason that high efficiency can only come with low stack temperatures. Chimneys only provide good draft with fairly high temperatures and with a consequent waste of much heat. Therefore, it seems natural to expect that this conflict of requirements will be solved by the use of a fan whose performance becomes increasingly better as the temperature falls and which gives absolute control of conditions of draft, rate of combustion, etc.

The limit of possibility in reducing stove capacity is set by the amount of heat it is possible to store in a given weight of brickwork with a reasonable increase in its temperature, for if too great a drop in the temperature of the brickwork be necessary more gas is required and this limitation must be carefully considered in any plan for reducing the stove capacity.

Three stoves with clean gas and high gas velocity may easily be better than four without those advantages, but whether it will ever pay to reduce the number of stoves below three may well be doubted until overwhelming proof is offered.

The **Gun-crete Company** has opened its offices in the McCormick Building, Chicago. The firm specializes in cement gun work for engineering, industrial and mining structures, and also introduces the Weber system of workingmen's cottages, which, being constructed by the cement gun process, are fire, water and vermin proof, and can be built at a price almost as low as ordinary frame buildings. Other specialties of the firm are the lining and fireproofing of mining shafts and the lining of coal bunkers, etc. Mr. Carl Weber is the president of the Gun-crete Company.

The question of pole supply is one of increasing importance as power lines are extended in the West. A study of the value of different Rocky Mountain woods for poles is the subject of Bulletin 67, U. S. Department of Agriculture.

Recent Chemical and Metallurgical Patents

Gold and Silver

Process for Canadian Silver-Cobalt Ores.—In a patent granted to Mr. **Oscar Dyckerhoff,** of Karlsruhe, Germany, the inventor states that "a rational, cheap process, which permits the recovery of all the silver and at least the larger part of the cobalt, nickel and arsenic (from Canadian ores) has hitherto been wanting." He proposes a process of chloridizing, as more nearly fulfilling conditions than does any process now in use. Ordinary chloridizing-roasting is a failure when metallic silver exists in an ore in large proportions. The inventor finds, however, that chloridization of such material is easily effected if clay or ordinary loam be added to the mixture of ore and salt. This combination of materials results in the formation, at a suitable temperature, of sodium silicate and chlorine or hydrochloric acid, which latter attacks the silver. The alumina of the clay absorbs the silver chloride. The reactions will ensue at a lower temperature if iron or other pyrite is added to the roast.

The effect of thus roasting at a temperature not to exceed 600 deg. C., is to convert cobalt and nickel into a form (mainly basic sulphate) soluble in dilute acid formed from the roaster gases in absorption towers. The arsenic exists as sodium arsenate, and the silver as chloride cobalt, nickel and arsenic are extracted by means of weak tower acid, and silver by sodium thiosulphate. Several successive roastings and leachings may be necessary in order to secure high extraction. Cobalt and nickel are recovered by precipitation as sulphides, while arsenic may be obtained by crystallization as sodium arsenate. Silver is precipitated from its solution as sulphide. The inventor gives the results of tests showing extractions of 99 per cent of the silver, 69 per cent of the nickel and cobalt, and 95 per cent of the arsenic. These results were obtained, however, only by four successive roasting and leaching operations. (1,085,675, Feb. 3, 1914.)

Roasting Precious-Metal Residue from Electrolytic Copper.—The slime or mud resulting from the electrolysis of copper generally contains gold and silver which are recovered by smelting. In order to remove copper as completely as possible before smelting, the residue should be roasted to render copper soluble in sulphuric acid. Mr. **Rowland T. Wales,** of Chrome, N. J., has patented a form of roasting furnace, specially adapted to the roasting of these residues. The material is first filter-pressed, then granulated and placed in the furnace in shallow trays. The roast is given at a temperature of 600 deg. F. The use of trays avoids stirring of the charge, with consequent loss of dust, and permits each portion to be roasted according to its needs. (1,085,831, Feb. 3, 1914.)

Lead, Copper, Nickel and Tin.

Making White Lead.—Roasting apparatus for producing white lead directly from lead ores has been patented by Mr. **Robert A. Stewart,** of Los Angeles, Cal. As shown in Fig. 1, the apparatus consists of a rotary roasting furnace 1, supplementary roasting chambers 23, and filter chamber 30. Suitable oil burners, consisting of an oil pipe, an air inlet and steam supply pipe, are placed in the rotary furnace at 14 and in the supplementary chambers at 28. The filter chamber has vertical partitions with movable, fabric-covered doors 36. A fan 33 induces a draft through the apparatus.

In the first operation in the drum, most of the lead is converted into oxide and carbonate, and the complete oxidation of the lead is accomplished in the supplementary chambers. The lead fume is retained in the filter chamber by means of the fabric-covered doors which are shaken from time to time by mechanism 44-46. The product may be removed through doors 37. (1,088,041, Feb. 24, 1914.)

Hydrometallurgical Copper Process.—The MacKay process for treating copper ores is one of a number of hydrometallurgical schemes proposed in the last few years. In a recent patent specification, Mr. **Henry S. MacKay** discloses the following features of his process. The ore is roasted at a low temperature in the presence of sulphur, whereby some of the copper

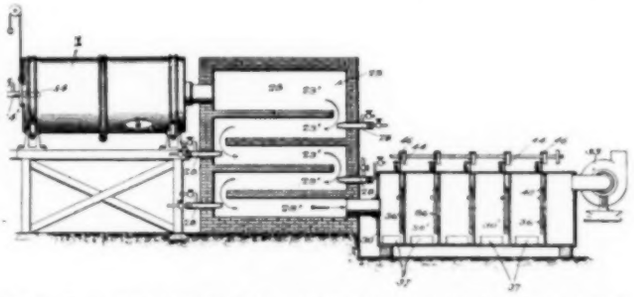


FIG. 1—APPARATUS FOR PRODUCING WHITE LEAD FROM ORES

is sulphated and rendered soluble in water. The ore is then leached with water, and the solution electrolyzed, copper being precipitated and sulphuric acid formed. The electrolyzed solution is then used as an acid leaching liquor for the extraction of the balance of the copper remaining in the ore as oxide or other form insoluble in water. The inventor finds a solution containing 2 per cent sulphuric acid and 2 per cent copper as sulphate is the best leaching liquor, and he preferably conducts his roasting and electrolyzing process in such a way as to maintain a standard solvent and electrolyte. Should he prefer to work with chloride instead of sulphate solutions, he adds salt in the roasting process, thereby obtaining a chloride electrolyte. (1,094,371, April 21, 1914.)

Sound Copper Castings.—A method of producing castings of copper and silver, free from blowholes and of high electrical conductivity, is disclosed in a patent granted to Mr. **Charles L. Spinney,** of Lynn, Mass., assignor to the General Electric Co. Two or more parts of B_2O_3 for every 100 parts of the metal to be cast are placed with charcoal or other inert material in the bottom of a crucible. Copper is then placed on this mixture and superheated to a temperature of 1300 deg. C. It is covered with charcoal. The molten mass is then stirred with a graphite rod, whereupon the molten boric anhydride rises through the mass and removes dissolved and combined gases. Cast copper thus prepared has conductivity as high as 90 to 95% Matthiesen standard. (1,091,542, March 31, 1914.)

Extraction of Nickel from Silicate Ores.—Garnierite ores contain nickel in the form of a double silicate of nickel and magnesia. Garnierite occurs usually in a matrix of serpentine, with magnesia micas and other minerals which it closely approaches in specific gravity, so that ordinary methods of water concentration are ineffective. Messrs. **Henry L. Sulman** and **Hugh F. K. Picard,** of London, Eng., have devised and patented a process for the recovery of nickel from garnierite by use of a mineral acid in such quantity as to act as a selective solvent of the nickel.

It is impossible to dissolve all the nickel from garnierite by acid treatment with dissolving also a large quantity of magnesia. But if the quantity of acid be limited to about 25 per cent of the weight of the ore, undue solution of magnesia is avoided. Weathered ores yield more magnesia to the solution than do freshly crushed ores.

The material is crushed to 30 or 50 mesh and treated with concentrated sulphuric acid (and some salt if desired—25 per cent acid and 10 per cent salt). Enough water may be added to form a paste, and the well mixed mass is heated to 120-130 deg. C. The soluble salts are then extracted by water, preferably hot and slightly acid, and the solution removed by a filter press. The nickel is precipitated as hydrate from this solution by stirring in powdered MgO. (1,091,545, March 31, 1914.)

Recovery of Tin.—A simple method of producing commercial tin from the mixture of finely divided tin and tin oxide resulting from the treatment of tin scrap, Mr. William Hoskins, of Chicago, Ill., proposes a special form of furnace. The shaft is equipped with rows of tuyeres at three different levels. It is filled with carbon to a level above the upper tuyeres, the fuel being maintained in a state of incandescence throughout, but at a higher temperature in the upper part of the furnace. The metal mixture is charged onto the incandescent mass in a reducing atmosphere where tin oxide is reduced, and all the metal flows to the bottom of the shaft where it is withdrawn. (1,088,422, Feb. 24, 1914.)

Metallurgical Furnaces

Metallizing Furnace.—The process of reducing an ore or oxide of a metal without fusion of the charge, known as metallizing, is receiving some attention as a possible means for producing an iron precipitant for copper solutions. A modified form of McDougall furnace is suggested and patented by Messrs. **E. P. Mathewson** and **Frederick Laist,** of Anaconda, Mont., as a suitable means of producing metallized iron, or "sponge" iron. According to the modification, a six-hearth furnace, for example, receives partly roasted ore, and completes the desulphurization and oxidation on the three upper hearths. On passing from the third to the fourth hearth, the ore receives an addition of crushed coal from a bin outside the furnace. The fourth and fifth hearths, where the reduction of iron oxide to iron takes place, are muffle hearths, suitably heated. Air is excluded from the three lower hearths and the final reduction takes place on the bottom hearth. The product is withdrawn from hoppers into which the ore flows from the bottom hearth. The roasting or oxidizing hearths are heated by oil burners, and the reducing hearths derive their heat from the admixed coal. Separate means are provided for withdrawing the different gaseous products from the upper and lower portions of the furnace. (1,094,802, April 28, 1914.)

Heat Conservation in Roasting Furnace.—According to a patent of Mr. **Utley Wedge,** of Ardmore, Pa., air heated by its passage through the rabble arms of the upper hearths of a furnace is delivered into the lower hearths through perforations in the rabble arms in those hearths. The heat derived from this air aids in maintaining combustion. (1,094,953, April 28, 1914.)

Furnace for Volatilizing Sulphur from Sulphide Ores.—In Fig. 2 is shown a modification of the Wedge furnace, in which the inventor has provided a construction for carrying on operations not easily performed in the ordinary furnace. Projecting upwardly from the hearths are annular ribs 11 which divide the hearths into inner and outer zones 12 and 13. The rabbles are adapted to work material in both zones. Different feed and discharge points are provided for the material going into and leaving each zone. Thus, for example, carbonaceous fuel may be burned in the outer zones and pyritic ore in the inner zones, the combustion of the carbon providing a reducing atmosphere in which sulphur may be volatilized. Steam may be introduced into those hearths where vaporization of the sulphur and oxidation of the metal is proceeding. This type of furnace is of interest in connection with the Hall process for roasting ores. (1,094,954, April 28, 1914.)

Lining Furnaces.—A method of lining and repairing furnaces of the open hearth type, while hot, is patented by **Mr. Byron E. Eldred,** of New York City. The method consists in projecting across a hot furnace, and to the point of use, a mixture of dolomitic lime and coal tar, or like substances, by means of a jet of products of combustion drawn from the stack of the furnace or other source. The material is protected against combustion of the binder in its passage across the furnace, and a dense, hard lining or patch can be readily made. (1,091,567, March 31, 1914.)

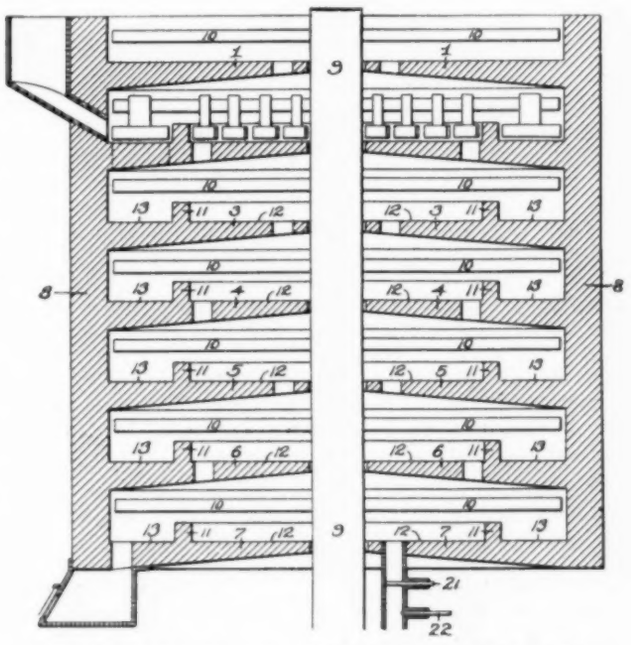


FIG. 2—FURNACE FOR VAPORIZING SULPHUR FROM SULPHIDE ORES

Alloys

Iron-Nickel-Copper Alloy.—The use of Monel metal in producing both high and low range alloys of this type, as well as their production directly from ore containing nickel, copper and iron in the form of sulphides, is the object of a patented invention of Mr. **Guilliam H. Clamer,** of Philadelphia, Pa. To make a low range alloy from ore, a matter is first produced, which is roasted and then reduced, yielding a homogeneous alloy of the three metals. (1,086,314, Feb. 3, 1914.)

Titanium Carbide.—The production of a metallic compound useful in removing occluded impurities from metals, is patented by Messrs. **Auguste J. Rossi** and **William F. Meredith,** of Niagara Falls, N. Y. Iron, titanium oxide and carbon are charged in a crucible of an electric furnace. Conditions of time and temperature are maintained which result in the production of the most possible titanium carbide throughout the bath. The mixture is then promptly cast to avoid formation of graphitic carbon. The product is useful as a purifying agent when added to molten metals. (1,094,022, April 21, 1914.)

Low-Carbon Ferro-Alloys.—The production of ferro-vanadium, ferro-uranium, ferro-chromium, etc., low in carbon, can be accomplished without using such

agents as silicon, aluminium, ferro-manganese, etc., according to a patent granted to Mr. **Warren F. Bleecker**, of Canonsburg, Pa. The inventor first reduces a suitable metallurgical mixture to produce a ferro-alloy high in carbon, and then mixes with it a calculated amount of unreduced oxides and slag-forming material, so that the excess carbon in the alloy first produced will reduce the oxides subsequently added. The result is an alloy unusually low in carbon. (1,094,114, April 21, 1914.)

Synopsis of Recent Chemical and Metallurgical Literature

Gold and Silver

Volatilization of Metals as Chlorides.—Reference to the process of volatilizing metals from sulphide ores by roasting with salt has been made in this journal, April and May, 1913, pages 220 and 230, respectively, and in February, 1914, pages 71 and 137. Readers who are interested in this metallurgical process will find a valuable contribution by Mr. STUART CROASDALE, in *Mining Magazine*, March, 1914. The author was one of the inventors of the process in this country about fifteen years ago. Last year Mr. Ben Howe, of Australia, announced a similar process. In the article here referred to, Mr. Croasdale compares at great length the details of his own and Howe's work, and shows remarkable concordance of results and similarity of experience on the part of both. In concluding his comparison Mr. Croasdale says:

"It seems to me that no more convincing testimony of the metallurgical soundness of this process could be given than this independent duplication of my work from beginning to end by Mr. Howe, ten years after my work was published. As will be seen, the greatest difficulty in both cases was mechanical, namely, securing a satisfactory apparatus for collecting the fume. I believe this can now be easily solved." The last remark refers to condensation by means of a high-potential electric discharge.

Benefits of Fine Grinding on the Rand.—According to a note in the *South African Mining Journal*, the results obtained by the new tube mill plant of the Knights G. M. Company afford a good example of the benefits to be derived from fine grinding. Five tube mills, 16 ft. 6 in. by 6 ft., have been added to the mill equipment, with the usual accessory apparatus for classifying the feed and amalgamating the discharge. The full equipment of 245 stamps and five tube mills is now capable of crushing 45,000 tons of ore per month. The value of the tubes lies more in increasing the extraction than in raising the tonnage.

In September last, before the tube mills were used, the tailing sent to the dump had a gold value of $0.487 per ton. In the last quarter of the year, following the use of tube mills, the value of tailing was reduced to $0.262. This represents a gain of $0.225 per ton. Milling cost was increased by about $0.12 per ton. Further reduction in tailing value and working cost is anticipated.

Sand Leaching at Oroya Black Range, Western Australia.—The monthly *Journal* of the Chamber of Mines of Western Australia for January, 1914, contains an article on this subject by Mr. W. B. CHOMLEY. The ore is an oxidized quartz in which a large proportion of the gold is deposited on the cleavage planes of the shattered crystals. Fine crushing is unnecessary, and leaching is simple. The sand treated by leaching comprises 54 per cent of the ore milled. Crushing is done by stamps through 20 and 14-mesh screens, one battery which receives the greater proportion of soft ore being equipped with 14-mesh screens. Recovery by amalgamation, both on plates and in grinding pans, amounts to 65 per cent of the total.

The arrangement of the leaching plant is unusual. Eight vats are arranged in two rows of four each, elevated on timber framework, 12 ft. from the surface. Two collecting tanks on movable platforms run on a triple line of rails, one above each row of treatment vats. A central launder extends between and above the two rows of treatment vats, and has branch launders through which the stream of pulp can be diverted to the collecting tanks wherever they may be.

When a vat is to be filled the collector is run to a point above it, and the stream of pulp is distributed into the collector. The slime overflows to the vacuum-treatment plant. The collector being filled, it is allowed to drain for ten hours, after which the charge is shoveled into the treatment vat below, through two doors in the bottom of the collector. After leaching, the treatment vat is discharged in a similar manner into cars beneath. The accompanying table gives details of treatment.

Charge—85 tons dry sand

Solution	% KCN	Time,	Hours	% KCN Coming Off
25 Tons Strong	0.25	Pumping Soaking Draining	5 8 19	.01 to .05
20 Tons No. 2 Sump	0.07	Pumping Soaking Draining	5 1 18	.06 to .08
25 Tons No. 3 Sump	0.03-0.05	Pumping Soaking Draining	5 1 18	.04 to .06
25 tons No. 3 Sump	0.03-0.05	Pumping Soaking Draining	5 1 18	.03 to .05
30 Tons No. 3 Sump	0.03-0.05	Pumping Draining Vacuum	5 21 4	.03 to .05
125 Tons			134	

No water wash is used, the weakest zinc-box tailing solution being applied for washing purposes.

The following is an average of costs for three months, and includes disposal of tailing.

	Cents per Ton
Labor and salaries	21.64
Power	4.01
Repairs	2.34
Assaying	2.01
Cyanide	16.32
Sundries	0.62
Disposal of residue	27.46
Total	74.46

The author calls attention to the value of allowing the charge to drain after each application of solution, thereby permitting oxygen to permeate the mass and aid in the solution of the gold. The practice of mixing some slime with sand is condemned on account of the practical impossibility of getting a permeable mass into which oxygen can enter.

Zinc, Lead

Preferential Flotation of Zinc Sulphide by the Horwood Process.—Our last issue contained a note on the adoption of the Horwood process by the Zinc Corporation, Ltd., Broken Hill, N. S. W. The information there given is supplemented by the present review of an article by Mr. E. J. HORWOOD, published in the *Proceedings* of the Australasian Institute of Mining Engineers, No. 12, 1913.

The material being treated is a mixed zinc-lead flota-

tion concentrate, too fine to be separated on concentrating tables. Hitherto it has been sold to zinc distillers as a low-grade concentrate. As it is not a desirable product, the producer is unable to obtain much value for the metals contained, particularly lead and silver. The importance of recovering these two metals in form suitable for lead smelting, as well as of raising the grade of the zinc content, is easily appreciated.

Being a preferential *zinc* flotation process, it offers advantages in treating slime concentrates produced from old dumps in which the lead particles have become slightly oxidized and are, therefore, not suited to a preferential *lead* flotation scheme. If the whole of the sulphides in any slime be first floated as mixed concentrates and then subjected to the Horwood process, a very high-grade zinc concentrate can be obtained, with a maximum recovery of the semi-oxidized lead and silver as a separate product.

A new feature of importance added last year was the preliminary washing of the mixed concentrates with clean water to remove soluble salts before roasting. This has resulted in diverting to the lead product a much higher percentage of the silver content, and has made possible an effective separation of lead and zinc with a lower degree of sulphatizing in the roast.

The Zinc Corporation's plant has a capacity of 500 tons per week. Pulp is thickened and then filter-pressed. The washing process mentioned takes place in the press, and the cake is finally discharged with about 10 per cent to 12 per cent moisture. It is then roasted in an Edwards furnace, in which the fuel consumption is 4.8 per cent of the ore charged. Coal is burned in six fireboxes arranged on both sides of the furnace. Draft is induced by a fan. The degree of sulphatization necessary for the process ranges about 30 per cent of the lead content.

The roasted product is then treated by flotation, whereupon the zinc floats and the lead and silver largely remain behind. Sulphuric acid consumed amounts to from 20 lb. to 27 lb. per ton. Eucalyptus oil is used at the rate of 1 lb. per ton, some being added to each of the ten boxes. Following are typical plant results:

	Zinc, Per Cent	Lead, per Cent	Silver, Oz. per Ton
Feed to roaster	39.0	16.0	18.0
Zinc concentrate	49.5	6.0	11.0
Lead residue	9.0	47.0	45.0

The financial result shows a net profit of about $15 per ton, considering the zinc concentrate as 71 per cent and the lead residue as 24 per cent of the original mixture.

Reverberatory Furnaces

The **utilization of the hot gases** escaping from a reverberatory furnace, as a source of power, is general practice in connection with copper smelting. In a paper published in the Mining Magazine, February, 1914, **Mr. Arthur E. Williams** contributes calculations bearing on the amount of heat that may thus be utilized in connection with oil-fired reverberatories. He considers the case of a modern furnace, 100 ft. long by 19 ft. wide, using crude oil of the following composition: 84.2 per cent C, 12.4 per cent H, 3.4 per cent O, having a calorific value of 19,296 B.t.u. A summary of his calculations leads him to the conclusion that a properly designed and operated waste-heat plant in connection with oil-fired reverberatories should yield in steam the equivalent of 50 per cent of the calorific value of the oil. Turbo-generators would yield 0.36 kw. or 0.48 electrical hp. for each pound of oil burned in the furnace.

Notes on Chemistry and Metallurgy in Great Britain

(By Our Special Correspondent)

Tungsten and Nickel High Speed Steels

A paper by Prof. J. O. Arnold and A. A. Reed on "The Carbides of Tungsten and Nickel Occurring in Steels" was recently read before the Institution of Mechanical Engineers.

In their introductory remarks the authors mentioned that although the importance of the subject might not be manifest to engineers, yet the work done by Abel and Deering on carbide of iron was a great cornerstone in the metallurgy of steel, and, therefore, for the benefit of engineers; because it appeared to them that advances in the metallurgy of steel had been followed by marked advances in mechanical engineering. It was not possible to acquire a complete knowledge of high-speed steels until the state of the existence of the carbides therein was well understood; and any importance which attached to the present paper was due to its dealing with tungsten, the essential element of high-speed steel, and with nickel, an equally vital component of the best structural steels.

The tungsten steels employed in the investigations contained from 0.67 to 0.73 per cent of carbon and from 2.38 to 26.29 per cent of tungsten. The carbides were separated by an electrolytic method, and the results showed that in the steel, with 0.7 per cent of carbon and 9.74 per cent tungsten, practically the whole of the iron carbide had disappeared, and the carbide of tungsten nearly corresponding to the formula WC had replaced it. That tungsten carbide should theoretically contain 93.88 per cent tungsten and 6.12 per cent carbon. In the case of steels containing 14.96, 21.15 and 26.29 per cent tungsten, the carbon falls and the iron rises as the tungsten rises, and this leads to the idea that a stable tungstide of iron exists with the carbide of tungsten.

In preparing ferro-tungsten at Sheffield University it had been found that the high specific gravity of this tungstide prevented the production of a homogeneous alloy in the crucible. When making pure 40 to 50 per cent ferro-tungsten from metallic tungsten and Swedish bar iron, it is always found that there is a fluid part which pours easily and a liquated portion at the bottom which can hardly be poured at all on account of its very high solidifying point. It adheres to the crucible in a pasty state, and when cold is only drilled with great difficulty by an especially hard drill which easily cuts very hard chilled white iron, and its hardness, therefore, must be nearly that of feldspar. In one melt the fluid portion of low solidifying point contained 38.2 per cent of tungsten and weighed about 14½ lb. The portion of higher melting point which remained in the crucible contained 52.3 per cent of tungsten, and its weight was about 13 lb. This latter metal has a composition approximately to the formula FeW. On account of this liquation it is quite impossible to prepare a homogeneous large bar of high-speed steel—for example, from a 2-ton ingot—the bottom of the ingot always contains considerably more tungsten than the top, and the larger the proportion of tungsten employed the greater is the difference between the tungsten content at the top and the bottom of the ingot.

Photo-micrographs of tungsten steels clearly showed the presence of three new components; namely, tungsten pearlite, tungsten cementite and tungsten hardenite. The pearlite is usually of a fine granular or minutely laminated structure, characteristic of the sorbitic variety of pearlite; the cementite was obtained in nodules, similar to iron cementite, by annealing the

tungsten alloys, owing to the segregation of the *W C* of the pearlite. The hardenite forms slowly and progressively from the pearlite. Beginning about 830 deg. C. the transformation is apparently complete at about 1150 deg. C.

The steels, which contained 14.96 and 21.15 per cent of tungsten with 0.7 and 0.73 per cent of carbon respectively, were easily filed when quenched at 860 deg. C.; when quenched at 1000 deg. C. they were hard, but could still be filed; but when quenched at 1190 deg. they were sufficiently hard to strip the files.

These facts go to prove the progressive formation of tungsten hardenite; and though these facts have been known for possibly twenty years to practical men, the authors believed that the scientific explanation was given for the first time in their paper.

With regard to nickel steels, the authors were of opinion that in ordinary manganese structural steels containing up to 7 per cent of nickel, the nickel is not combined with the carbon, but forms alloys with the iron and the manganese, and that a definite alloy of iron and nickel is formed corresponding to the formula Fe_7Ni, a substance which is possibly one of the most remarkable in the metallurgy of iron.

Sir Robert Hadfield suggested they should make a set of nickel alloys, as free as possible from carbon, silicon and manganese. He predicted that alloys of nearly pure iron and nickel would show a critical mechanical point at something between 11 and 15 per cent of nickel. Experimentally the tests fulfilled the prediction, but the effect of carbon and silicon and higher manganese on such alloys was to produce surprising ductility, giving 11 per cent elongation, 46 per cent reduction of area, a bending angle of 167 deg.; a yield point of 39 tons and a maximum stress of 88 tons per square inch for the 13 per cent nickel steel.

A nearly pure 7 per cent nickel-iron has almost the same mechanical properties as an iron containing 33.5 per cent nickel, namely, a yield point of 30 tons, a maximum stress of about 38.5 tons per square inch; an elongation on 2 in. of about 30 per cent, and a 37 per cent reduction of area. A curve of tenacity for nearly pure nickel iron with observation points at 7, 13 and 33 per cent nickel the greatest stress is shown at 13 per cent, which corresponds to the formula Fe_7Ni. The consequent existence of a definite alloy of iron and nickel having that composition is extremely possible.

Scottish Shipbuilding

During February 27 vessels with a tonnage of 55,687 were launched from Scottish yards, making a total of 46 vessels and 86,500 tons for the two months, out of which 32 vessels of 71,354 tons were built on the Clyde.

Although new orders placed are far from numerous there is no immediate probability of work slackening in the yards.

Company Meetings

Sir W. G. Armstrong Whitworth & Co.

Annual general meeting, April 18. A total dividend for the year of 2/ 6d. per share on the ordinary shares was declared and £56,559.14.9 went for dividends on the preference shares, the total sum thus absorbed being £568,122.4.9, leaving a balance of £323,716.2.3 to be carried forward to next year's accounts.

Mason & Barry.

Annual general meeting, April 27. A dividend of 7/- per share was declared, the same as last year. After placing £20,000 to reserve for contingencies, and £1,000 as usual to Staff Pension Fund, the balance carried forward was £15,550, as against £25,148 last year.

Consolidated Malay Rubber Estates.

Annual general meeting, April 23. Total dividend of 50 per cent for the year was declared, and it was announced that the cost of production had been materially lowered.

Bell's United Asbestos Co.

Annual general meeting, April 20. The ordinary dividend declared is the same as for the last four years, namely, 12½ per cent; £5,000 goes to reserve, as against £15,000 in the previous year, 1912, and more than £3,000 more is carried forward.

Brush Electrical Engineering Co.

Annual general meeting, April 20. The chairman announced that the business of the company had improved very satisfactorily. It was proposed to reduce the preference shares to a nominal value of 7/- and the ordinary shares to a nominal value of 5/- each, with a view of being in a position next year to pay a dividend on the reduced capital.

Market Prices

April, 1914

	£ s. d.
Aluminium ingots, ton lots	86. 0. 0
Alum, lump, loose, per ton	£5.5.0 to 5.10. 0
Antimony, Star Regulus, ton	28.10. 0
Borax, British refined, crystal, cwt	17.10. 0
Copper sulphate, ton	21.15. 0
Caustic soda, 70% ton	10. 0. 0
Copper ore, 10% to 25% per unit	10/9 to 11. 3
Ebonite rod, lb	4. 6
Hydrochloric acid, cwt	5. 0
India-rubber, Para, fine, lb	2.11½
Mica, in original cases, medium	3/6 to 6. 0
Petroleum, Russian spot, gal	9¼
Quicksilver (Spanish) bottle	7. 0. 0
Sal-Ammoniac, cwt	2. 2. 0
Sulphate of ammonia, ton	12. 3. 9
Sulphur, recovered, ton	5.10. 0
Shellac, cwt.	3.15. 0
Platinum, oz., nominal	9. 5. 0
Tin ore, 90%, ton	£96.0.0 to 98. 0. 0
Zinc, Vielle Montagne	25.12. 6

Copper opened £64.15.0 and soon hardened, being £65.18.9 by the 8th. After the holidays it was lower at £64.5.0 (14th) and £64.17.6 (15th), falling away to £64.7.6 (20th) and, after a short recovery, to £63.5.0 (24th). Since slightly higher and closing £64.0.0.

Tin opened £172, but fell away till it touched £167.7.6 (8th). After the holidays it opened £166, weakening for a few days, then regaining to £166.5.0 on the 17th. It declined sharply later, reaching £161.10.0 on the 22d and £155.10.0 on the 24th, closing £156.15.0.

Lead opened £18.10.0 and rose to £18.15.0, which price was maintained from the 7th to the 16th. It then fell to £18.10, remaining so till the 23d. On the 24th again rose to £18.15.0, and by the 27th had reached £19.15.0 and closes £19.5.0.

Hematite has been steady at 61/- and closes same price.

Scotch pig opened 56/- but rose smartly to 57/6 (7th), after the holidays (14th) it opened 57/3 and then fell slowly to 56/7½ on the 22d, recovering to 56/10½ on the 23d. Closes 57/6.

Cleveland opened 50/- and rose to 51/6 by the 7th, thence fell to 51/3, which was maintained till the 14th; on the 15th 50/10½ was registered, the price then fell off to 50/7½ on the 22d, after which a recovery to 50/10½ on the 23d. It closes 51/4½.

Differences

Higher	£ s. d.	Lower	£ s. d.
Aluminium, ton	1. 0. 0	Copper sulphate	15. 0
Shellac, cwt	3. 6	Sul. of Ammonia	3 6
Scotch pig	1. 6	India-rubber	0½
Cleveland	1. 4½	Tin ore	11. 0. 0
		Copper	15. 0
		Tin	15. 0. 0

The American Institute of Mining Engineers will hold its autumn meeting in the week from Monday, Aug. 10, to Saturday, Aug. 15, in Salt Lake City, Utah.

The Lavoisite System of Generating Oxygen

For the art of autogenous welding, and the cutting of steel, there are at present four commercial methods of generating oxygen. Probably the user of oxygen for such purposes is most familiar with bottled oxygen, as produced by the Linde process.

Second in volume to the bottled oxygen as produced by the Linde process is that produced by the decomposition of water, which is also distributed in bottles, under high pressure, and is commercially known as electrolytic oxygen. Electrolytic cells are now also installed in manufacturing plants, making it possible for such manufacturers to produce their own oxygen, thereby doing away with the inconvenience incident to the use of bottled oxygen.

The production of oxygen by the use of chlorate of potash and manganese dioxide is now widely known, and there are several small installations of such plants throughout the country.

The Lavoisite system, though not so well known, is in fact a much older method of producing oxygen than any of the previously mentioned systems. Its introduction into general use for the generation of oxygen, in connection with the art of oxy-acetylene welding, has been slow, for the reason that up to within a comparatively short time a practical commercial generator had not been designed that would work successfully in connection with this method of generating oxygen, and in this respect the Lavoisite process is similar to a great many other chemical processes, the commercial efficiency of such depending entirely upon a design of apparatus that will carry out the chemical reactions in an efficient and reliable manner.

Lavoisite, so called in honor of the celebrated French chemist Lavoisier, whose name is so intimately associated with oxygen, is a specially prepared calcium hypochlorite, which, when put into solution, a suitable and proper amount of catalyzer being added and the solution simultaneously being heated to 90 deg. C., rapidly generates oxygen of a purity that is unsurpassed by any of the processes that are now in commercial use. Tests from plants in operation showing the oxygen from Lavoisite to be practically 100 per cent pure.

In the early days of the art of oxy-acetylene welding the purity of the oxygen was not given much thought. It soon became evident, when the users of oxygen became familiar with the art of autogenous welding, that the purity of the oxygen was of the greatest importance, as the results obtained by the use of pure oxygen against impure oxygen were such that even a slight difference of only 1 per cent would show inferiority in the quality of the weld, the time to accomplish the work, the amount of oxygen and acetylene used also being affected. This is most pronounced in the cutting of steel, where a difference of only a fraction of 1 per cent will greatly affect the time of cutting and quantity of oxygen used.

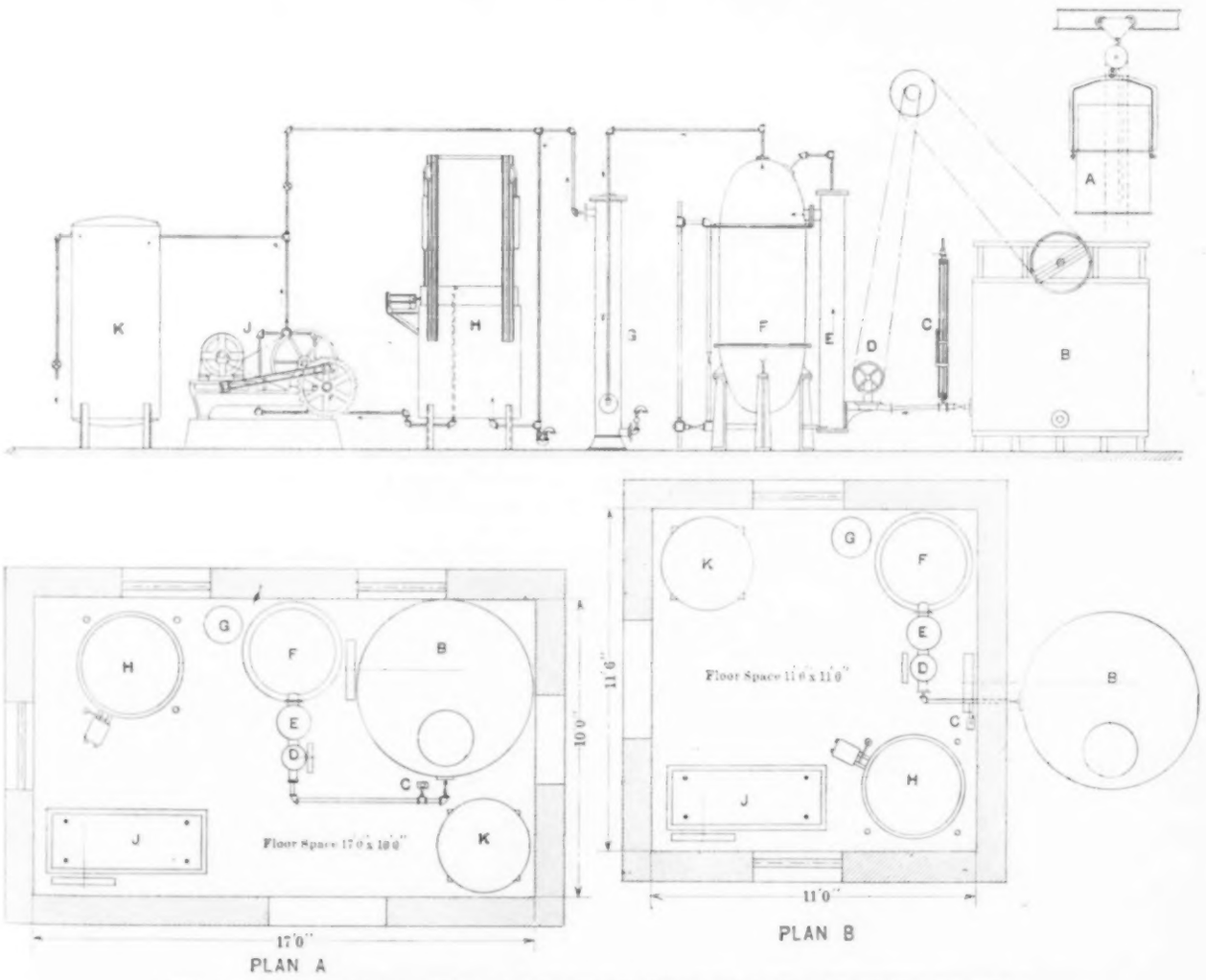


FIG. 1—PLANS AND VERTICAL SECTION OF LAVOISITE OXYGEN-GENERATING SYSTEM

A loader, B mixing tank, C indicator, D pump, E heater, F generator, G scrubber, H gas-holder, J compressor, K pressure-tank.

The operation of the Lavoisite generator is best described in U. S. patent No. 1,031,017, covering this apparatus, the efficiency of its operation depending upon the regulation and direction of the flow of the Lavoisite solution through the apparatus.

Lavoisite is shipped in large hermetically sealed steel drums, holding about 800 lb. of Lavoisite. The Lavoisite is charged into tank *B*, shown in Fig. 1. A loading device of special design is furnished for lifting the drums from the floor and conveying them to the mixing tank *B*. The loader is equipped with doors that open

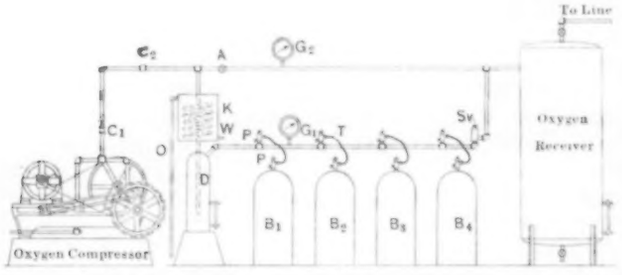


FIG. 2—METHOD OF COMPRESSING LAVOISITE OXYGEN INTO CYLINDERS FOR PORTABILITY IN USE THROUGHOUT CUSTOMER'S PLANT

A globe valve on receiver line, B_1, B_2, B_3, B_4 oxygen bottles, C_1 vertical ball check valve, C_2 horizontal swing check valves, D separator, K condenser, G_1 high pressure gauge, G_2 low pressure gauge, O water overflow, P pressure valves, Sv safety valve, T copper tubes, W water inlet.

downwards upon releasing a catch, thereby discharging the contents of the drum into the mixing tank. Sufficient water is then added to bring the mixture up to the required consistency. An indicator *C* shows the required amount of water to be added, according to the weight of Lavoisite placed in the mixing tank. This indicator does away with any necessity of calculation on the part of the attendant in charge of the plant as to the proper strength of the Lavoisite liquor in the mixing tank, which it is essential to keep within a few degrees of the correct gravity.

Mixing tank *B* is equipped with an agitating device to assist the Lavoisite in going into solution, and during the generation of oxygen this agitator is slowly, yet constantly, stirring the contents of the mixing tank.

The liquor passes from the mixing tank to the pump *D*, which forces it into heater *E*. Connected with pump *D* is a small pump for simultaneously forcing catalytic liquor into the heater *E* in the correct proportion to Lavoisite liquor being pumped by pump *D*.

The heater *E* is equipped with a thermostat which controls the amount of live steam admitted to heat the liquor coming from the pump, so that regardless of the speed of the pump over a given range, the temperature of the heater remains constant. The Lavoisite and catalytic liquor enter at the bottom of the heater and are brought quickly up to a temperature of 90 deg. C., when oxygen is rapidly generated; it is here necessary that the liquor in the heater travel in the same direction as the liberated oxygen.

About 90 per cent of the reaction takes place in the heater. The oxygen liberated passes through a pipe connected to the top of the heater to the generator *F*. The partly finished liquor passes through a pipe in the side of the heater to the generator *F*, within which the reaction is completed, the spent liquor overflowing through the pipe shown at the side of the generator. The passage of the liquor through the generator is in reverse direction to that of the oxygen; consequently the liquor that has been unacted upon, in its passage through the heater, comes into more intimate contact with the catalyzer in the generator *F*, with the result that the liquor leaving the generator is entirely depleted of its oxygen content.

The oxygen leaving the generator passes through the scrubber, or cooler, *G*, where it is reduced in temperature, and any moisture mechanically carried over from the generator is condensed out. From the scrubber the oxygen passes to a gas-holder *H*, the bell of which is counterbalanced.

The rise and fall of this bell actuates a tank switch, which automatically cuts in and out the compressor *J*. The pressure of the oxygen up to the compressor is slight, in practice about equal to a column of water 2 in. high. The compressor raises this pressure up to any desired point, usually 75 to 100 lb., and stores the oxygen at this pressure in the pressure tank *K*.

A pipe shown at the left of the pressure tank conducts the oxygen to any point where it is desired to use it, the usual reducing valves being placed on outlets from this pipe to reduce the pressure suitable to the torch being used.

The capacity of the Lavoisite generator is very flexible, the quantity of oxygen produced being controlled by the speed of the pump *D*, which is fitted with a cone pulley, making it possible to generate oxygen from 50 cu. ft. per hour to 500 cu. ft. per hour. This is of great advantage to the welder, as it enables him to keep pace with his work by simply speeding up the apparatus, doing away with the necessity of having numerous or large tanks of oxygen under high pressure.

Tests of the oxygen have shown a purity as high as 99.9 per cent. The purity of Lavoisite oxygen is such, in fact, that for the cutting of metal it is not necessary to run the pressure above 50 lb. for all ordinary work. If, however, it is desired to compress to a higher degree than that usually carried in the pipe lines, an outfit such as is shown in Fig. 2 is used, which allows the consumer to compress the oxygen into bottles, up to a reasonably high pressure, 300 to 400 lb., for portable and local use about his plant.

Plans of typical installations of Lavoisite plants are shown in Fig. 1. It will be seen to be possible to place in a building 11 ft. square a generator capable of producing 500 cu. ft. of oxygen per hour.

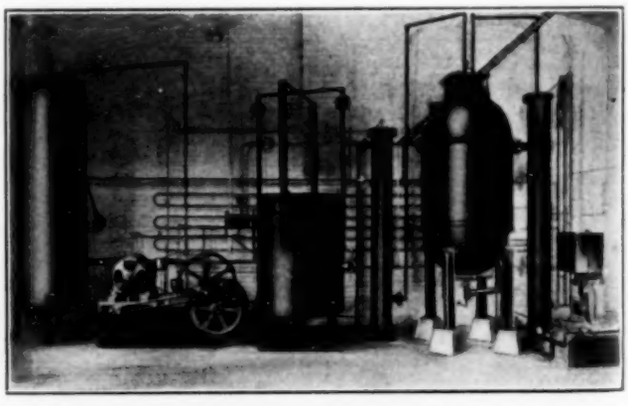


FIG. 3—LAVOISITE OXYGEN-GENERATING PLANT, 50 TO 500 CU. FT. OXYGEN PER HOUR

The operation of the generator is automatic and continuous, so long as it is supplied with liquor from the mixing tank *B*, and this tank can be charged from time to time without affecting the generator. In shutting down it is necessary only to stop the feed pump *D*, when the generation of oxygen ceases.

Fig. 3 is a view of an installation with a capacity of from 50 to 500 cu. ft. of oxygen per hour. The mixing tank is here not shown, it being placed on the outside of the building. The pipe shown projecting through the

wall at the right in the photograph connects with mixing tank. The illustration clearly shows the several parts of the apparatus. There are, however, numerous pipes shown in the photograph having nothing to do with the Lavoisite generating outfit, they being water pipes for heating and water supply to the welding shop.

Lavoisite is noncombustible, so that large quantities can be stored without danger. This makes it possible for the user to have at his command his supply of oxygen in a convenient form to meet the varying demands of his daily needs. Sufficient quantity of Lavoisite is usually charged into the mixing tank in the morning to last throughout the day; this is a matter of a half hour's work, so that it is not necessary to employ additional labor to do this work, the ordinary shop helper being fully capable of charging the Lavoisite plant.

Lavoisite is sold by the Castner Electrolytic Alkali Company under an agreement with the purchaser that every pound of Lavoisite will produce 1 cu. ft. of oxygen, when used in the generators designed for its use and operated according to instructions given. The apparatus, and all pertaining thereto, are manufactured by the Castner Electrolytic Alkali Company, Niagara Falls, N. Y. Some of the plants now installed by this company are capable of generating upwards of 5000 cu. ft. of oxygen per day of ten hours.

The **New York Section** of the **American Electrochemical Society,** in its meeting on May 22, elected Dr. Colin G. Fink chairman for the coming year and reelected Mr. H. B. Coho secretary and treasurer. A vote of thanks was extended to the retiring chairman, Mr. Lawrence Addicks, now in Douglas, Arizona, for the extraordinarily effective work done by him for the society during the past year.

Detachable Link-Belt is the subject matter of a very interesting and freely illustrated book just issued by the Link-Belt Company of Chicago as Section A of Catalog 110. Detachable link-belt was invented by William D. Ewart in 1873, and from that time the original has been manufactured solely by the Ewart and Belmont Works of the Link-Belt Company.

Feed-Water Difficulties.—"The problem of suitable feed water, although less serious in New England than in other parts of the country, is still a trouble maker in many plants. The formation of scale and the softening of water are due to the simplest of chemical reactions, and by an analysis of the water a competent chemist can readily predict in advance the proper amounts of suitable chemicals necessary to prevent scale or corrosion, and not only save the company the expense of frequent boring and replacing of tubes, but obviate the necessity of attempting to force heat through the same substance with which many of your steam lines are insulated. The service of a chemist," said Mr. Carl F. Woods, secretary of the firm of Arthur D. Little, Inc., of Boston, when recently speaking before a group of street railway operators, "would prevent the purchase of a special compound at $1,000 a year which consisted of 97 per cent water and 3 per cent molasses, or obviate the necessity of purchasing a mixture of soda ash, tannin and water under a brand name at 8 cents a pound when the principal ingredient can be obtained for 1 cent a pound."

Lead and Zinc in Kansas and Arkansas.—The value of the lead and zinc produced at mines in Kansas in 1913 was $1,343,432, compared with $1,680,744 in 1912, according to J. P. Dunlop, of the United States Geological Survey. In Arkansas the value was $55,120 in 1913, compared with $106,014 in 1912.

Accelerators for Gasoline Gas Plants

BY C. J. BARTON

The laboratory in which the writer worked was equipped with a gasoline gas machine. In the summer no trouble was experienced in securing a good gasoline gas for the laboratory but in winter the gas became so weak that it would hardly burn. The apparatus consisted of a large drum-shaped low-pressure air-compressor which compressed air to about 10 in. of water. The drum was driven by a large weight which was periodically wound up by hand. The gasoline was kept in a large underground tank outside of the laboratory. The gas machine was a surface machine, the air from the compressor passing over the surface of gasoline in the tank then through a pressure regulator, where the pressure was reduced to about 1 in. of water, then into the gas line.

It was almost impossible to do any analytical work with the weak gas obtained in the winter time so the writer made what was termed "accelerators" to enrich the weak and almost unburnable gas into a good gasoline gas.

The "accelerators" consisted of a wide-mouthed heavy glass bottle with a two-hole rubber-stopper fitted. The bottle would hold, when full, about a pint of gasoline. The rubber stopper was fitted with two pieces of glass tubing bent at right angles at their upper extremity, to fit into the gas hose.

One glass tube was about 6 in. in length and led the weak gas from the line down into the bottle to the surface of the gasoline, the other tube was just long enough to reach through the rubber stopper and carried the enriched gas through a short length of gas tubing to the burner. The bottle if filled with gasoline in the morning would enrich the gas for a half day or so. The object of having the one piece of glass tube longer than the other was to make it possible to follow the surface of the gasoline down into the bottle as it was evaporated. This also furnished a means of regulating the amount of the enrichment the weak gas received. A bulb was blown in the glass tube leaving the "accelerator" in which glass wool was inserted to prevent the flame following back into the bottle from the burner on turning the gas off. One of these accelerators was made for each burner used.

It was noted on using these "accelerators" that the more volatile and lighter constituents evaporated first and left a heavier portion behind in the bottle which amounted to from 50 to 60 per cent of the original volume of the gasoline. This residuum very much resembled kerosene in appearance but would not burn when poured on the cement floor. It could be completely vaporized, however, if warmed to the boiling point by setting the "accelerator" in hot water or on a radiator. Ordinarily in using the "accelerator" when 40 to 50 per cent of the gasoline had evaporated the remainder was thrown out and more gasoline was put in.

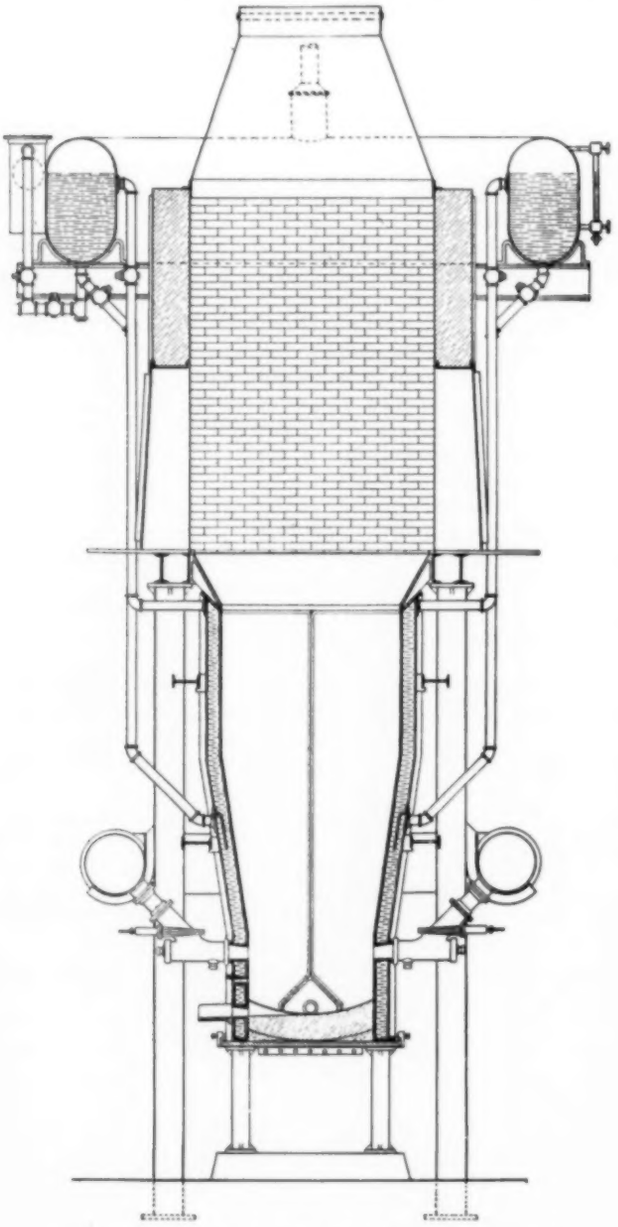
These "accelerators" have been in use about three months and have proved a valuable accessory to an independent gasoline gas plant.

The **Munning-Loeb Co.,** of Matawan, N. J., have just issued Bulletin 500 on their line of plating and industrial brushes. The book contains information on brush manufacture from the making of the block, selection and origin of stock to the final finish. Operating characteristics of the several brush materials and their adaptability to certain classes of work are briefly discussed.

The Vaporization of Jacket Water

Catalog No. 12-F, just issued by the Colorado Iron Works Co., of Denver, Colorado, and entitled "Some Details as to Smelting Practice and Equipments," contains an exceedingly interesting chapter on the Vaporization of Jacket Water, which deserves the full attention even of experienced smelting men. We herewith reproduce this chapter practically in full.

In many localities the matter of jacket water supply is a very serious one, either from absolute scarcity, expense of pumping, or cost, where it has to be purchased. This led the Colorado Iron Works to a serious study of the problem, and their solution of it, the Nesmith patented system of vaporization, not only reduces the amount required to one-eighth of that ordinarily necessary, but also provides what experience has proved to be the best method of cooling even where water is abundant.



NESMITH PATENT JACKET WATER VAPORIZER

Some smelter operators run so much water through their jackets that it is discharged at a temperature much below the boiling point. Their reason for this is primarily to obviate the close attention otherwise necessary and they also think that they are on the safe side and that the cooler jackets are run, the better. This is a mistaken idea. Aside from the fact that an important loss of heat, which must be compensated for by additional fuel, results from such practice, the life of the jackets is much greater when they are run with the discharge as near the boiling point as possible. Both cast iron and steel have greater tensile strength and a higher elastic limit at 400 degrees Fahrenheit than at lower temperatures, and the hotter jackets can be kept, the more durable they will be. More cast-iron jackets are broken and steel jackets damaged by running an excess of cold water through them than in any other way. The sudden admission of cold water to a cast-iron jacket which has been running hot is especially liable to cause its failure.

As indicated by its name, the principle upon which this system is based is to take advantage of the latent heat of vaporization of water. The magnitude of the saving thus effected is plainly apparent in the following figures.

The heat absorbed by one pound of water at 212 degrees F. in passing into steam is 966 British thermal units, the application of this amount of heat being necessary to produce the change from liquid to gaseous state. In cooling jackets in the ordinary manner, the heat units carried off by the water equal the number of degrees difference in temperature between the feed and discharge, and assuming the feed water temperature to be 62 degrees and the discharge 212 degress, the maximum possible heat abstraction of one pound of water is 150 units.

If now the pound of water be completely vaporized, there will be 966 units added to the 150 units, or a total of 1,116 units of heat utilized. This shows the amount of water necessary when vaporized to be, $150 \div 1116 = 0.1344$ or 13.44 per cent. of that necessary where the water is discharged as water at the boiling point. The results in actual practice are better than this, for the discharge of the water at the boiling point has been assumed, and this is a condition which is never even closely approached, owing to the danger of supplying insufficient water.

The water obviously can not be boiled in the jackets, but our system provides means whereby full advantage is taken of the latent heat of vaporization without danger of overheating any jacket—in fact, with absolute assurance of all jackets being continuously and evenly cooled.

The vaporizing system operates in a manner similar to the hot-water system ordinarily installed in residences, in which the water back in the range corresponds to the water jackets of the blast furnace. The arrangement of piping is similar and circulation of the water is maintained in the same way.

For use with this system the jackets are the same as with the ordinary overflow system except that the pipe connections are larger in order to reduce friction. The pipes lead to a closed drum surrounding the furnace, the feed pipes being connected to the bottom and the discharge pipes entering the drum slightly below the water level. The tendency of the hot water to rise and cold water to find the lowest level maintains a very effective circulation and any boiling takes place in the drum, not in the jackets, because the water in the jackets is under sufficient head to cause the boiling point to be several degrees higher than in the drum which is only under atmospheric pressure. A gauge glass indicates the water level, an automatic float valve controls the admission of water to replace that evaporated, and a vapor escape pipe provided with an exhaust head conducts the steam formed to the atmosphere.

A smelter superintendent, after operating a furnace provided with this vaporization system for several months, recently gave the following information on jacket water consumption:

"The furnace proper is using about 90 gallons per hour. I have not had time to figure this back to gallons per square foot of jacket area, but as the jacket area on this furnace is large, it should show a very low water consumption per square foot. * * * It is without question the best water system for a blast furnace I have ever seen. All jackets are always at a uniform temperature; with a proper float valve absolutely no attention is required. In blowing-in we never think of the water, while with the old-style overflow system it generally keeps one man busy to keep from burning a jacket. In places where water is scarce a more elaborate condenser on the exhaust head would save nearly all the water. Another very good feature is the fact that different furnace men have their own ideas about jacket water—one wants to run his water cool or cold and another hot with no two jackets the same, while with this system it absolutely cuts out the 'personal element' and the jackets must have a decided benefit from the constant temperature."

Jacket water consumption is usually determined in gallons per minute, to avoid large figures, but as is evident from the above quotation, gallons per hour would be a more convenient measure of comparison were the vaporizer system in universal use. However, on the customary basis, the exposed jacketed surface of the above furnace being 414 square feet, the water consumption was 0.00362 gallon per minute per square foot of jacket area. This was a copper matting furnace. Another user of the vaporizer system, operating a lead furnace, reported a water consumption of 0.0113 gallon per minute per square foot jacket area, but this vaporizer was run without an automatic float valve and loss of some water was consequently unavoidable.

Comparison of the above with published figures for Cananea, 0.616, and for Granby, 0.31, shows an actual saving far above the theoretical, and brings out strongly the magnitude of the waste of water in cooling jackets in the ordinary way. At Granby, the discharge from the upper jackets was the feed for the lower jackets, which accounts for the large economy shown over Cananea, the amount of water used at Cananea being about what is ordinarily required.

The jacket water is commonly run into settling ponds where it is cooled and reused, but there is great loss from evaporation and seepage, to say nothing of the expense of pumping. The installation of the vaporizing system will cost less than settling ponds and it entails no continual expense, as for pumping. In addition, it provides cooling means in every way superior to the ordinary overflow method.

Radium Prices.—At the Senate hearing on radium Mr. Thomas F. V. Curran, of Placerville, Colo., who is engaged in mining and shipping carnotite ores, stated that he sold his product to Armet de Lisle, Nogent sur Marne, France, where he obtained the best prices. Payment is made on the basis of hydrous radium bromide, $RaBr_2.2H_2O$, delivered, at the rate of 100 francs ($19.30) per milligram per ton, for ore containing from 6 mg to 10 mg per ton; over 10 mg per ton a bonus of 20 per cent is paid. As the ore contains from 3 mg to 4 mg $RaBr_2.2H_2O$ per ton, the price received ranges from $60 to $80 per ton of ore. Mr. W. L. Cummings, another shipper from Placerville, stated that he was paid $2 per pound for the uranium oxide contained, f.o.b. Placerville. On 2 per cent ore, which is average shipping grade, the price would come to $80 per ton.

Electric Furnace Regulator

In the interesting American Electrochemical Society paper of Mr. Robert Turnbull, of Welland, Ontario, Canada, noticed on page 333 of our May issue, a number of important principal advantages were mentioned which can be obtained in electric furnace operation by the use of nearly all kinds of regulators.

Among special advantages which may be derived from the use of the Thury regulator, Mr. Turnbull mentioned the following:

1. The damping devices employed only permit the regulator to act, when it is necessary for it to do so. Sudden changes and changes of short duration will not put the regulator in action. This in itself is very important, as not only is there a great saving on the wear and tear of the electrode hoisting mechanism, motors for driving same, etc., but the current is kept more uniform by avoiding the regulation of such changes.

2. This regulator can handle any amount of power for regulating purposes, without in any way impeding the free play of the controlling mechanism, which is only in contact with the power mechanism for exceedingly short periods of time and is otherwise perfectly free.

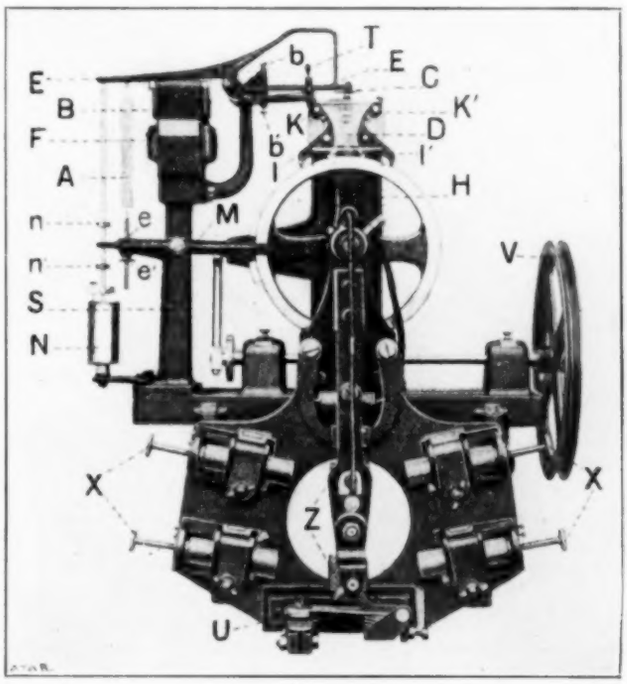


FIG. 1—THURY REGULATOR

3. The regulator requires no attention whatever, and with the exception of the oiling of the moving parts once a week, can be practically ignored.

4. Great variations of current can be automatically obtained by use of this regulator.

5. The intermittent action of the contact-making arm is important, as the regulation is thereby prevented from being carried too far in one direction and a high load factor is assured at all times and uniform current on the furnace.

The Thury regulator is the invention of Mr. R. Thury, of Geneva, Switzerland. It was first applied to electric furnaces in 1901 by Dr. Paul Heroult to his first steel furnace:

The principal parts of the regulator (Fig. 1) are the fixed coil F, the movable coil B, the rocking lever E, the spiral spring A, the dashpot or damping device

N, the abutment knives C, the tappets K, the pawls I, the tappet wheel H, the copper contacts Z, the carbon contacts, and the short-circuiting device U.

When the current operating the furnace is at its normal value, the rocking lever E is held in equilibrium by the pull of the spring A, and the magnetic force of the fixed coil F acting on the movable coil B. The

5. A current transformer T, operating the solenoid of the regulator.

One cycle of operation in this system would be as follows:

We will suppose that the current flowing through the secondary bars to the furnace is above the normal. The current flowing from the secondaries of the trans-

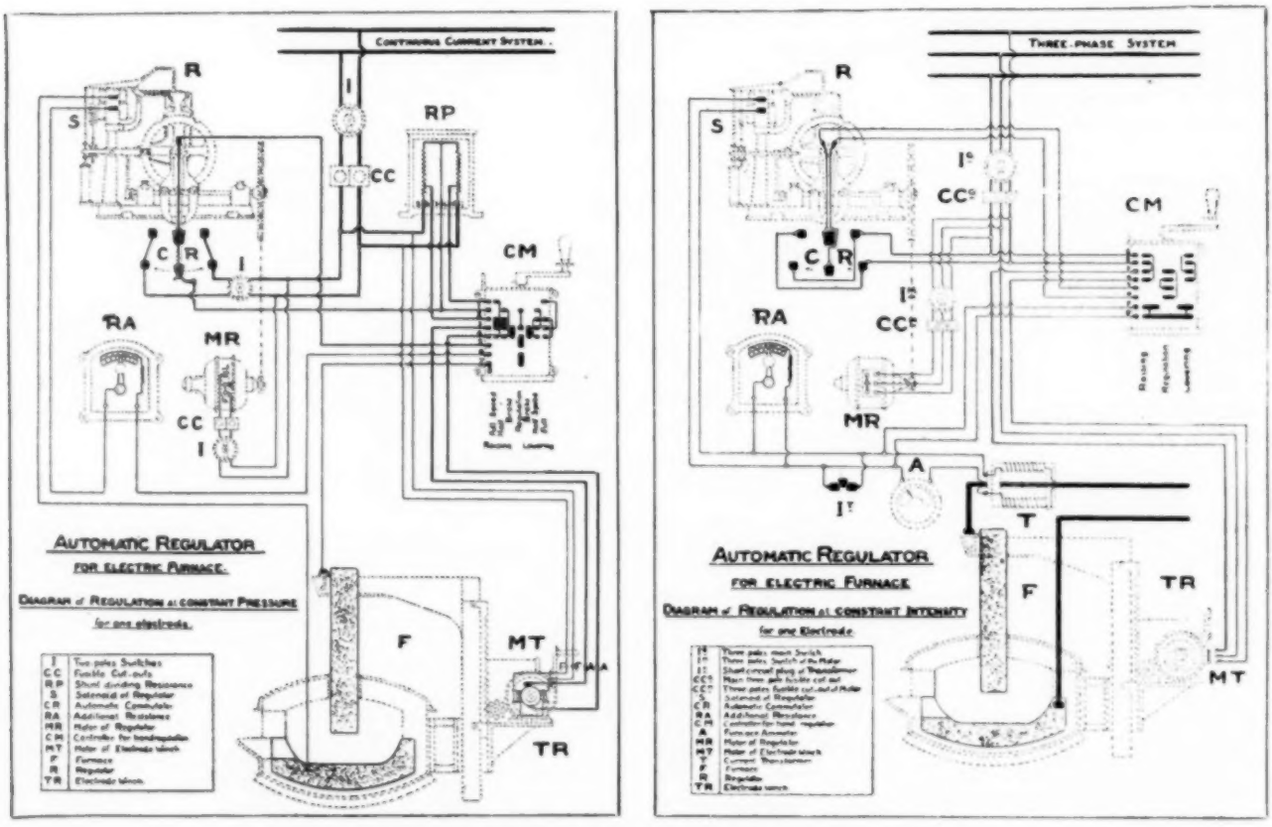


FIG. 2—AUTOMATIC REGULATOR SYSTEM FOR ELECTRIC FURNACES

knives fixed to the rocking lever in this position swing clear of the tappets K. When a variation of sufficient duration occurs on the furnace, the movable coil will either be pulled to or pushed away from the fixed coil, depending on whether the current has decreased or increased on the furnace, and the rocking lever will swing either the one or the other of the knives against one of the tappets K. The tappet is pushed from its position, releases the pawl I which falls into the notch in the tappet wheel. The arm carrying the copper contacts, which is fixed to the tappet wheel shaft, will swing either to the right or to the left, bringing the copper contacts against the carbons, and current will immediately flow to the terminals of the winch motor.

The Cuenod Company of Geneva, Switzerland, who now control the manufacture of the Thury regulator, have added to the regulator a series of apparatus, which taken together form a very complete regulator system. This system can best be described by a reference to Fig. 2.

It consists of the following:

1. A winch T R for raising and lowering the electrodes.

2. A double-throw switch with carbon contacts CR, worked by an automatic Thury regulator R, which starts, stops, and reverses the motor MT as desired.

3. An adjustable resistance RA, permitting a modification of the current on the furnace, for instance, from 2000 to 10,000 amperes, by steps of 1000 amperes.

4. A controller CM for hand and automatic control.

former T to the solenoid of the regulator, will also be above normal, and the increased magnetic force will push the movable coil against the pull of the spring until the knife C will encounter the tappet K, pushing the latter out of its position, thereby releasing the pawl I. This pawl falls into the notch in the tappet wheel H which in turn brings the copper contacts Z on the lever, in contact with the carbon contacts, and the current immediately flows to the winch motor through the controller CM. When the current on the furnace has been brought back to its normal value, the regulator will cease to act and the lever carrying the copper contacts will remain in its central position, until again called upon to act, through a variation of current in the furnace.

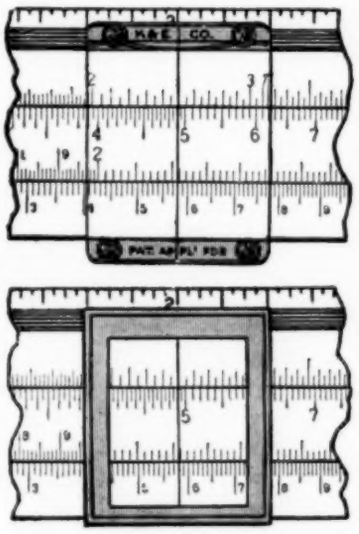
The mineral output of California for the years 1905 and 1912, with comparative estimate for 1913, is shown in the table below, compiled by F. McN. Hamilton, State Mineralogist:

Substance	1905	1912	1913
Petroleum	$9,007,820	$41,868,344	$46,000,000
Gold	19,197,043	19,713,478	20,000,000
Cement	1,791,916	6,074,661	8,000,000
Copper	2,650,605	5,638,049	5,500,000
Crushed rock and granite	2,070,607	5,473,928	6,000,000
Brick	2,273,786	2,940,290	3,000,000
Borax	1,019,158	1,122,713	1,250,000
Natural gas	102,479	940,076	1,250,000
Silver	678,494	799,584	800,000
Quicksilver	886,081	866,024	700,000
Other minerals including asphalt, clay, lead, marble, mineral earths, salt, tungsten, zinc, etc.	3,391,238	6,035,238	7,500,000
Totals	$43,069,227	$91,472,385	$100,000,000

The Latest Slide Rule Improvement

A new patented indicator or runner for slide rules, called the "frameless," has just been perfected by Keuffel & Esser Company. Every figure on the rule is clearly visible at all times, there being no side pieces to the metal holder of the glass indicator, and therefore, nothing to hide any of the figures on the rule (see Fig. 1). This is one of the most important improvements in slide rules.

Many times, after setting the old-style indicator or runner, the user would find that he could not read the result because important figures were hidden by the frame or holder of the glass (Fig. 2). Frequently two, and sometimes, four or even six, eight or nine figures would be thus hidden; causing more or less inconvenience and uncertainty in reading the slide rule. The new "frameless" indicator or runner entirely obviates this difficulty.



FIGS. 1 AND 2—SLIDE-RULE INDICATOR—NEW STYLE, UPPER PICTURE; OLD STYLE, LOWER PICTURE

Thousands of engineers and scientists in every profession and industry, as well as contractors, builders, architects and merchants have found the slide rule invaluable as a time and brain saver in quickly and accurately making a great variety of calculations. In its various forms, it has been adapted by Keuffel & Esser Company to the needs of practically every line of work, including all branches of engineering, as well as chemistry. A new slide rule, especially adapted to the needs of the chemist, was described in our March issue, page 201.

Compressing Air and Gases

One matter more than anything else has held back the introduction of compressed air for many industrial purposes. This has been the cost of compressing the air. The first cost of the compressor itself is high. Moreover, expensive foundations must be installed. Then the compressor and the receiver occupy much valuable floor space. The services of an experienced engineer are required to operate it. Taking all this together, it is clear that the overhead expense is a big item and becomes especially serious where the air is only required intermittently. Further, the power is an important consideration. All this explains why managers of industrial plants have been hesitating to introduce compressed air for purposes for which it would have been otherwise very useful.

A decisive advance has been made by the Nash Engineering Company, of South Norwalk, Conn., in the design of the Nash hydroturbine. These pumps are capable of delivering air or gas at 15 lb. pressure on continuous twenty-four-hour service with absolute reliability and high efficiency and it would seem that the Nash hydroturbine, because of its simplicity, low first cost, and low installation expense, small space occupied, absolute reliability and high efficiency, opens up a large field for the use of compressed air.

There is no longer any necessity of having long lines of air piping, with big losses due to leakage and air friction. A small Nash compressor, belted off a line shaft or direct connected to an electric motor can be located at the place the air is used, and started up by the operator of the machine when wanted.

Heretofore it has been the usual practice, where some of the work required high-pressure air, to install a compressor giving a pressure of 60 to 100 lb. and reduce the air needed at low pressure by throttling or by means of a reducing valve. Oftentimes 75 per cent of the work in the plant can be handled with entire satisfaction with a pressure not exceeding 10 to 15 lb. It is needless to say that there is no economy in compressing air to 100 lb. and then reducing it to 15 lb. for use.

The Nash hydroturbine offers a practical solution of this problem. A hydroturbine is installed of a capacity sufficient for the entire plant for both high and low-pressure air. This pump is arranged to maintain a pressure of 10 to 15 lb., depending upon the pressure required for the bulk of the work in the plant. If some air is required at 60 to 100 lb., a small compressor is installed, taking its air from the delivery side of the Nash hydroturbine and boosting it to the desired pressure. The compressor for the high-pressure air will be small, only one-half to one-eighth the size that would otherwise be necessary, the size depending on the amount of high-pressure air required.

The space occupied will be less. There will also be a saving in power, both for the high-pressure air, as well as the low-pressure air, because the Nash hydroturbine is very efficient. The air is used at the pressure to which it is compressed, and two-stage compression is used for the high-pressure air. The compression in the

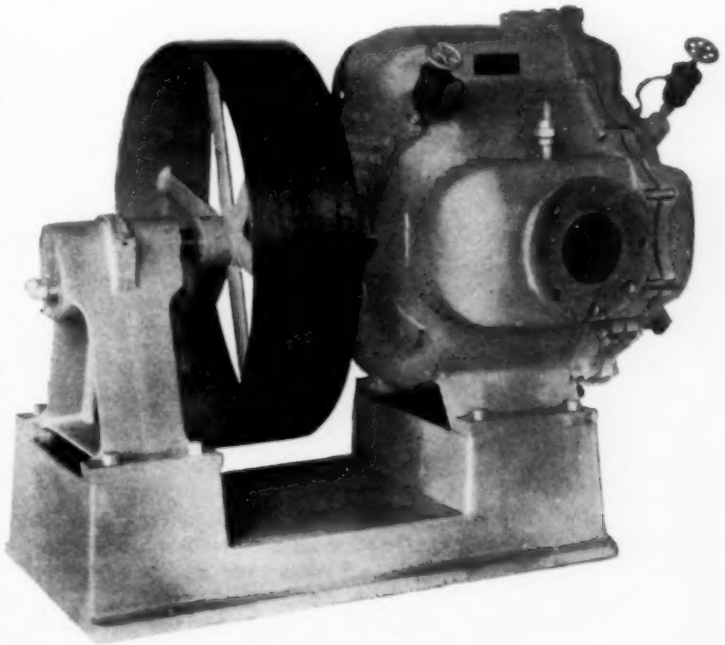


FIG. 1—HYDROTURBINE

Nash is isothermal, therefore no intercooler is required.

The Nash Engineering Company, the exclusive manufacturers of air compressors and vacuum pumps working on the hydroturbine principle, have been building these pumps for about four years. Fig. 1 shows the latest single-stage design of their hydroturbine. It differs from the first pumps put on the market only in minor details, such as the additional outboard bearing, position of the outlet and inlet piping, etc.

The extreme simplicity of the pump is best illustrated by Fig. 2, wherein all the principal parts are shown. The case, head and wheel constitute the pump.

Fig. 3 is a cross-section showing the principle of operation. The pump consists of a casing and a head, surrounding a wheel with projecting blades, which is rigidly mounted on a shaft, supported by bearings in the head and casing. The casing is elliptical in section. The rotor, the periphery of which is a true circle, clears the short diameter of the elliptical casing by 1-16 in. to 3/32 in., depending upon the size of the pump, so that there is no possibility of the blades striking, even if the bearings become worn.

Above and below the narrow part of the ellipse, ports are cut through the casing, and similar ports are cut through the head, exactly opposite. These communicate with cored passages, connected to the inlet and outlet pipes.

Water, or other liquid, is introduced into the pump. The rotor is revolved by power applied to the shaft, carrying the water with it. The water, however, follows the casing due to centrifugal force.

This means that twice in a revolution it recedes from and surges back into the rotor, due to the elliptical form of the case. As the water goes out of the rotor it draws air in through the ports to fill its place. This air is carried around with the rotor, and is compressed and delivered through the outlet ports by the re-entering water. Each half of this elliptical case is really a separate pump. As the outlet and inlet ports are diametrically opposite, the pressures on the rotor, due to compression, are exactly balanced, thus relieving the bearings of all side-pressure.

Since the blades of the rotor are passing the outlet ports continuously, the air is delivered without pulsation.

The blades of the rotor are reinforced by two shrouds, as will be seen in Fig. 2, leaving just enough space between the hub to register with the ports. The steel shaft is a drive fit in the rotor, and is fastened securely by two pins going clear through the interior hub and shaft.

The shaft is supported by three bearings, one each in the head and casing, and one forming the outboard

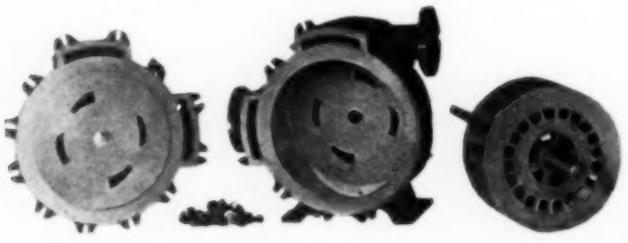


FIG. 2—CONSTRUCTION OF HYDROTURBINE

bearing. The pump and outboard bearings are dowelled to the case, and the head flange is recessed and fitted to the casing, so that exact alignment will be preserved if the pump is taken apart and reassembled.

The bearing bushings are made of best grade of babbitt, containing about 85 per cent. tin. They are fitted so that they can be tapped in with a light mallet, and are prevented from turning by means of a lug cast in the bushing, which fits in a recess in the containing casting.

The outboard bearing is ring-oiled. The bearings in the head and casing are lubricated with graphite grease, as this is more reliable than oil in the presence of water. The grease is fed to the bearings by means of compression cups of special design, located near the top of the pump. The air pressure is admitted to the top of these cups to balance the pressure, so that the grease will flow to the bearings readily.

The outlet and inlet pipes are equipped with standard flanges. The design is such that the head can be removed and the rotor taken out without disturbing the piping.

All parts of the pump are made absolutely interchangeable.

It is apparent from the above that there is nothing about these pumps to get out of order. In fact, they are practically indestructible with any reasonable care. In addition to being absolutely reliable, they are very efficient, occupying small space, and are quiet in opera-

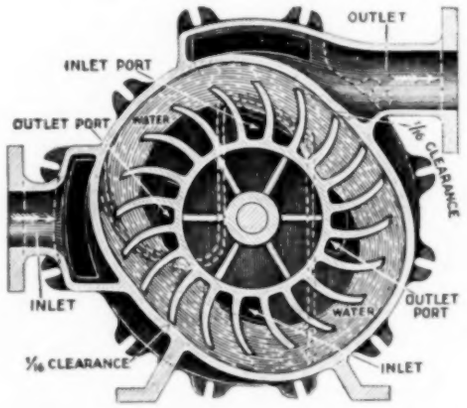


FIG. 3—PRINCIPLE OF OPERATION OF HYDROTURBINE

tion. They are so simple that anybody can operate them. All that is necessary is to keep them supplied with lubricant.

The standard pumps such as used for compressing air are made of cast iron. Where mild acid gas is to be handled, sufficient protection can be secured by keeping the contained water slightly alkali. Where the particular conditions of the plant would result in the corrosion of the cast iron the pumps are made of any other suitable metal or alloy.

While we have described in the above the use of the Nash hydroturbine for compressing air or gases it may be added that it is also a powerful and efficient vacuum pump.

Monel metal, the natural copper-nickel alloy, is finding a wide variety of uses, among which is the manufacture of non-corrodible window screen cloth.

American Ingot Iron in England.—The American Rolling Mill Company of Middletown and Zanesville, manufacturers of American ingot iron, have invaded England by closing up two contracts under which the Shelton Iron, Steel & Coal Company, of Stoke-on-Trent, and Richard Johnson & Nephew, Ltd., of Manchester, secure the right of using the American Rolling Mill Company's method. The capacity of the American works of the latter company has increased from 40,000 tons of sheet in 1906 to 166,000 tons at present.

Large Metal Value in Utah.—The value of the mine output of gold, silver, copper, lead and zinc in Utah in 1913 was $44,858,210, compared with $42,922,302 in 1912, according to Victor C. Heikes, of the United States Geological Survey. The production of gold was valued at $3,565,229, a decrease of $700,622, or 16.42 per cent, compared with 1912. The largest producer of gold was Salt Lake County, which yielded $1,777,214, against $1,788,730 in 1912. The West Mountain or Bingham district, in Salt Lake County, produced $1,774,815, or 49.78 per cent of the total output. The Tintic district, in Juab and Utah Counties, ranked first in gold output in 1912, but was second in 1913, with $1,412,462, or 39.61 per cent of the total.

Prospects of Electric Zinc Smelting in British Columbia

From the *Daily News* of Nelson, B. C., we learn that there is an excellent chance of the Johnson electric zinc furnace being tried out on a commercial scale in Nelson or Trail, British Columbia. As the Johnson furnace and process are so well known to our readers, we simply report that a party consisting of Messrs. Byron E. Eldred, W. McA. Johnson and James Struthers, all of New York, and A. E. Barlow, of Montreal, visited Nelson and Trail on May 4, 5, 6 and looked carefully over the ground.

The plan is for the erection of a ten-ton furnace plant at a cost of $75,000, and the Nelson Board of Trade passed unanimously a resolution:

"That the Board favor and support the proposal that a grant be made by the Dominion government to the Johnson Electric Smelting, Inc., of $25,000 on completion by them of a demonstration plant of 10-ton capacity in West Kootenay ready for operation, and a second government grant of $25,000 upon a successful commercial demonstration of the operation of same by a continuous thirty-day run under the oversight and to the satisfaction of a representative of the government."

Digest of Electrochemical U. S. Patents

PRIOR TO 1903.

Arranged according to subject-matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

Electroplating (Continued)

175,134, March 21, 1876, Charles S. Minchew, of Taunton, Mass., assignor to Reed & Barton, of same place.

Relates to electroplating, especially upon soft metals, such as britannia ware, with other metals, and calls the process "impulsio-plating." The article is first made chemically clean by boiling in caustic alkali, then mechanically cleaned with sand, brushes and water. It is then wired in running water. It is then connected as cathode in a hot alkaline solution of oxalate of soda and a strong current passed. It is then quickly transferred to the plating-bath and the desired deposit made in a hot solution under pneumatic pressure. After the deposit, the article is washed and dried, and then subjected to a temperature of 180 deg. F., which is said to "drive in" the deposit.

215,463, May 20, 1879, Nathaniel S. Keith, of Brooklyn, N. Y.

Relates to the electrolysis of lead and increases the conductivity of the electrolytes by adding to solutions of lead salts, similar salts of more electropositive metals, such as of sodium, magnesium, aluminium, etc. The more nearly saturated, the higher is the conductivity.

273,467, March 6, 1883, Alexander Classen, of Aachen, Germany.

Relates to electroplating from solutions of oxalates, which are made as follows: to solutions of chlorides or sulphates of the metals to be plated are added solutions of potassium or ammonium oxalate to form double salts; ammonium oxalate is then added in excess to the hot solution, and later a small quantity of sodium carbonate may be added. For nickel, the following proportions are given: five hundred grams nickel sulphate and eight hundred grams of ammonium oxalate are dissolved in ten liters of water. The deposit is made from a heated solution, using an anode of carbon or platinum.

300,035, June 10, 1884, William Henry Walenn, of Islington, county of Middlesex, England.

Relates to electroplating copper, etc., at temperatures near the boiling-points of the solutions, in closed vessels, the solution being under pressure of about one atmosphere, or less. For copper, an electrolyte is made as follows: thirty-two ounces, avoirdupois, of cyanide of potassium, and one ounce of neutral tartrate of ammonium are dissolved and diluted to one imperial gallon. The solution is then charged with copper from an anode until about three ounces per gallon is dissolved, using a large anode and small cathode. To this solution is added ammonia cupric oxide, until the solution turns slightly green in color. The ammonia cupric oxide is made by adding caustic potash to copper sulphate, boiling the solution, removing the precipitate and dissolving it in strong ammonia water of 0.880 specific gravity, to saturation. The deposits of copper are said to be adhesive and ductile, and not to separate when the plated sheets are hammered, etc.

330,149, November 10, 1885, Gaston Pilet and Clément Carry, of Besançon, France.

Relates to electroplating palladium, and describes an electrolyte made of chloride of palladium containing ten grams of metal; five liters of water; one hundred grams of phosphate of ammonium; five hundred grams of phosphate of sodium; ten grams of benzoic acid, which may be omitted, but is said to bleach the deposit, and renders the deposit more striking upon iron or steel. The bath may be used cold to produce a steel color deposit; if heated to from 40 deg. C.-50 deg. C., the color is whiter. A feeble current is preferred. Another electrolyte is also described.

331,104, November 24, 1885, William H. Winslow, of Brooklyn, N. Y.

Relates to electroplating articles with iron, and then converting the iron coating into magnetic oxide. An electrolyte containing twenty parts of salammoniac to one hundred parts of water is electrolyzed with two wrought iron plates for anode and cathode. After the electrolyte is suitably charged with iron, the cathode is removed and the article to be plated inserted. The plating must be carefully conducted; about one hour's time being considered sufficient. The iron-coated article is then removed and heated in an atmosphere of superheated steam in a muffle at about 1000 deg. F. to 1200 deg. F. for from five to seven minutes, which converts the iron coating into magnetic oxide. It is said to have a steel-bluish-black color, hard, non-corrosive and very attractive in appearance. Refers to a copending application which became patent No. 331,105.

331,105, November 24, 1885, William H. Winslow.

Relates to articles having a coating of magnetic oxide of iron, produced by the process described in his copending application which became patent No. 331,104. The patent describes the production of an electrotype having a magnetic oxide of iron printing face.

Personal

Mr. **Lawrence Addicks** has resigned as superintendent of the Chrome, N. J., refinery of the U. S. Metals Refining Company, and is now in Douglas, Ariz., in an advisory capacity, to study the possibilities of leaching at the various Phelps Dodge properties and direct the large-scale experiments under way at present at Douglas.

Mr. **E. L. Messler,** formerly with the Jones & Laughlin Steel Company, as general superintendent of their Pittsburgh coke ovens, blast furnaces and ingot mold foundry, and for the past two years as assistant to the president of Riter-Conley Manufacturing Company, is no longer associated with the latter firm He has as yet not decided what line of work he will follow. Mr. Messler's present address is care of the Duquesne Club, Pittsburgh, Pa.

Mr. **William P. Baer,** formerly of the Sprague Electric Works of the General Electric Company, has joined the forces of the Trumbull Electric Mfg. Co., and will represent them in part of the Middle West and also a part of the Southern territory.

Mr. **George H. Garrey** has severed his connection with the American Smelting & Refining Co., and allied companies, as their chief geologist, and has opened an office as consulting mining geologist and engineer at 115 Broadway, New York.

Mr. W. R. Hulbert, of the Goldschmidt Thermit Company of New York, presented recently a fascinating lecture on thermit and its latest developments in railway shop practice before the Railway Club of Pittsburgh. The paper is printed with illustrations, together with the extended discussion which followed, in Vol. 13, No. 3, of the official *Proceedings* of the Railway Club of Pittsburgh.

Mr. **Henry E. Jacoby** announces the removal on May 1 of his offices to larger and more spacious quarters at 95 and 97 Liberty Street, where he will continue to specialize on chemical machinery and equipment of all kinds, including a much larger and more improved line of vacuum drying apparatus.

Mr. J. B. Mendenhall, formerly with the National Lead Company, is now southern representative of the Great Western Smelting & Refining Company.

Mr. **J. Malcolm Muir,** business manager of *Metallurgical and Chemical Engineering,* was married on May 14th to Miss **Lida C. Kelly** in Louisville, Ky. Mr. and Mrs. Muir are now enjoying a honeymoon in the sunny South, somewhere in the Blue Ridge Mountains.

Dr. **Joseph Struthers,** formerly secretary of the American Institute of Mining Engineers and editor of Mineral Industry, has accepted the office of second vice-president of the Johnson Electric Smelting, Inc., controlling the American rights of the Johnson electrothermic process for the treatment of zinc ores and zinciferous lead and copper ores, with offices at 18 East Forty-first Street, New York City.

Mr. W. P. Thornton, who has been with the Riter-Conley Mfg. Co. for many years in various capacities, has resigned to accept a position with the Treadwell Construction Co. During the erection of the eight Gary blast furnaces of the United States Steel Corporation he was the engineer in charge for the Riter-Conley Company.

Mr. **Alois von Isakovics,** of Monticello, N. Y., lectured on May 8 before the Institute of Arts and Sciences and the Senior Organic Students of Columbia University on "perfumes and flavors, their chemistry, synthesis and practical application." He was introduced by Professor M. T. Bogert and after the lecture a vote of thanks was extended to Mr. von Isakovics on motion of Dr. C. F. Chandler.

Obituary

Herman Frasch died on May 1 at his home in Paris at the age of sixty-two. Herman Frasch was born in a small town near Stuttgart, Wuerttemberg, the son of the burgomaster of the town. Leaving the gymnasium at Halle, at the age of sixteen, he apprenticed himself for a time to a pharmacist. He came to the United States in 1868 and found his first employment in the laboratory of Prof. John M. Maisch at the Philadelphia College of Pharmacy. A little later he established a laboratory of his own, in which he made his first invention of importance, completed in 1876 a new process for refining paraffin wax. This he sold to a subsidiary of the Standard Oil Company. In 1885 he bought an oil field in Canada, started the Empire Oil Company, and soon invented the process for eliminating the sulphur which before that time had prevented the use of the oil for illuminating purposes. This invention was purchased by the Standard Oil Company, whose chief chemist Frasch then became. After serving as chief chemist of the Standard Oil Company for over twenty years, Frasch turned his attention to extracting sulphur from the beds underneath the marshes of Louisiana. This was accomplished by sinking shafts down to the bed through which the sulphur was melted by superheated steam and forced up by the pressure generated. This novel industry is in the hands of the Union Sulphur Company, of which Frasch was president. In 1912 the Perkin Medal was awarded to Herman Frasch by the Perkin Medal Committee, consisting of representatives of the Society of Chemical Industry, the American Chemical Society and the American Electrochemical Society. A report of the proceedings at the presentations of the Perkin Medal to Herman Frasch will be found in our issue of February, 1912, which contains a full account of his life and his lifework.

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Dr. **Walter Feld,** a prominent German gas engineer, died on March 15th. Among his most notable inventions is a centrifugal gas washer for the purification and cooling of blast-furnace and producer gases, for the recovery of metal dust from gases of lead and copper smelting plants, for the extraction of tar from coke-oven gases, etc. A very interesting new process of Dr. Feld relates to the extraction of ammonia and hydrogen sulphide from coal gases and the direct production of ammonium sulphate therefrom without the use of sulphuric acid. Two interesting articles by Dr. Feld on these subjects appeared on pages 395 and 436 of our volume X (1912).

Book Review

The Copper Handbook. A Manual of the Copper Mining Industry of the World. By **Walter Harvey Weed.** Vol. XI., 1912-13. Price $5. Houghton, Mich.: Walter Harvey Weed, 1914.

Under the able editorship of Horace Jared Stevens, the Copper Handbook had grown from a modest book of 300 pages, covering Michigan mines, till the tenth edition (1910-11) had become a manual of the copper industry of the world and a volume of 1902 pages. It had become recognized authority and its account of the doings of the copper mining companies was accepted as impartial and fearless when Stevens died on April 12, 1912.

Fortunately his untimely death did not mean the end of the Copper Handbook. It was acquired by Mr. Walter Harvey Weed a year after Stevens' death and the eleventh volume has just been issued. It may be said to be fully up to the standard set by its founder.

The first part (989 pages) gives detailed notes on all copper mines in North America, alphabetically arranged, and is followed in the second part (39 pages) by a very useful geographical index in which the mines are grouped by countries, states, counties, and mining districts. Part III. (285 pages) gives detailed notes on foreign companies. Part IV. (47 pages) contains statistics and part V. (53 pages) a list of dead mining companies.

"As in former editions, any gold, silver and lead mines producing copper as a byproduct has been included, practically making the book a mining manual, at least for America." As a reference book, in the arrangement of its contents, the present volume shows considerable improvement over former editions.